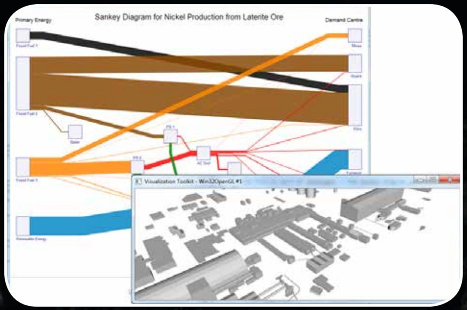
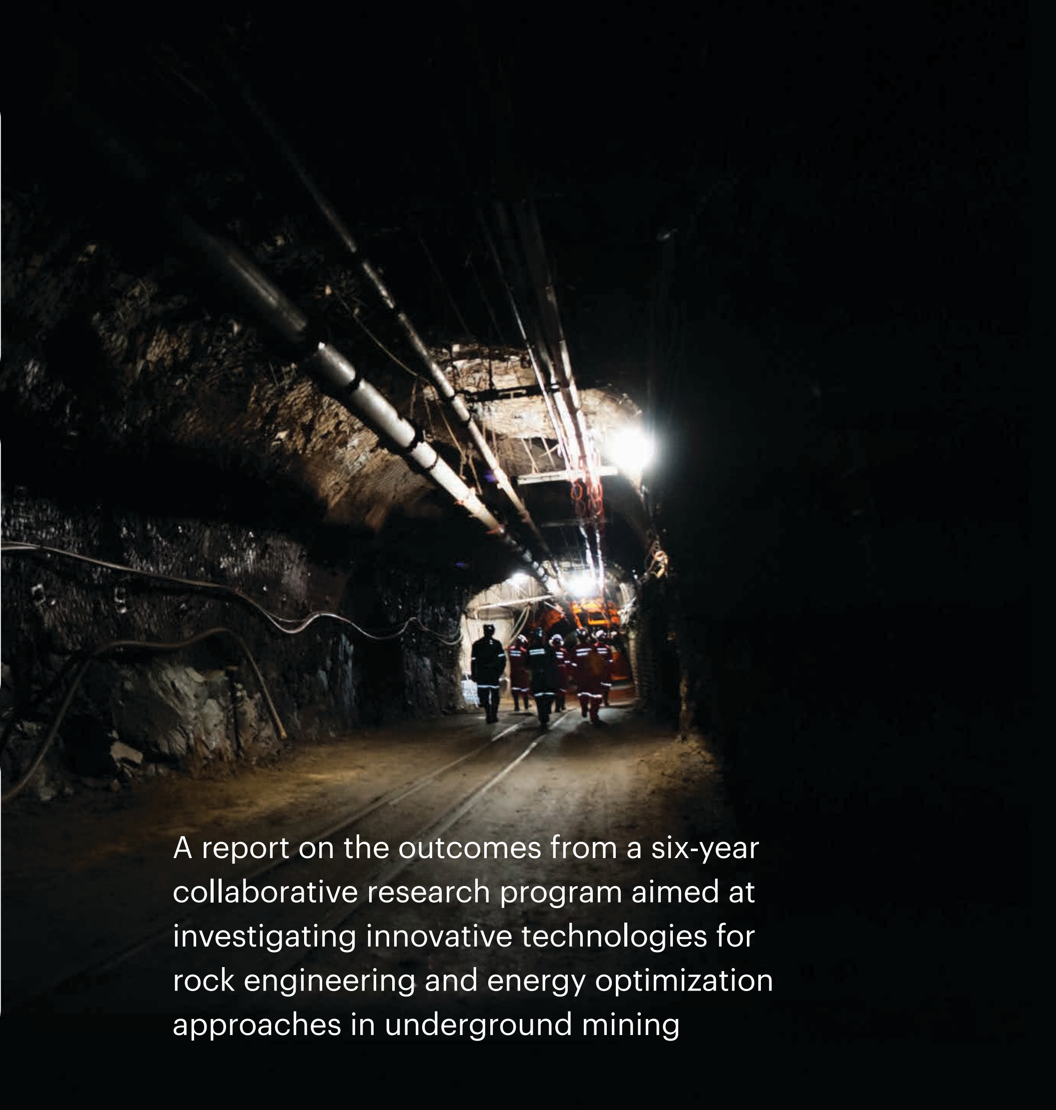
S.U.M.I.T.
PROGRAM JOURNAL

Smart Underground Monitoring & Integrated Technologies for Deep Mines

A Laurentian University-led and CEMI-managed R&D Report 2017





A report on the outcomes from a six-year collaborative research program aimed at investigating innovative technologies for rock engineering and energy optimization approaches in underground mining

CEMI
Centre for Excellence in Mining Innovation

Laurentian University
Université Laurentienne

Editors: Mr. Damien Duff | Dr. Peter K. Kaiser | Ms. Shannon Katary

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EDITORIAL TEAM:

Mr. Damien Duff dduff@cemi.ca
Dr. Peter K. Kaiser pkaiser@laurentian.ca
Ms. Shannon Katary skatary@cemi.ca

PRINCIPAL RESEARCH INSTITUTIONS:

Laurentian University, Queen's University, University of Toronto

PRINCIPLE RESEARCH CONTRIBUTORS:

Laurentian University
Dr. Peter Kaiser pkaiser@laurentian.ca
Dr. Ming Cai mcai@laurentian.ca
Dr. Dean Millar dmillar@mirarco.org
Dr. Richard Smith rssmith@laurentian.ca

Queen's University
Dr. Mark Diederichs diederim@queensu.ca
Dr. Jean Hutchinson hutchinj@queensu.ca
Dr. Steve McKinnon steve.mckinnon@queensu.ca

University of Toronto
Dr. Bernd Milkereit bernd.milkereit@gmail.com
Dr. Kaiwen Xia kaiwen.xia@utoronto.ca
Dr. Paul Young paul.young@utoronto.ca

University of Waterloo
Dr. Maurice Dusseault mauriced@uwaterloo.ca

Carleton University
Dr. Claire Samson claire.samson@carleton.ca

University of Alberta
Dr. Doug Schmitt doug@phys.ualberta.ca

University of British Columbia
Dr. Erik Eberhardt eric@eos.ubc.ca

Mira Geoscience
Dr. John McGaughey johnm@mirageoscience.com

DESIGN & PRODUCTION:

Katrina Teimo, Dovetail Communications Inc. www.dvtail.com

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Mining Leadership

With the formation of the Stevenson Commission in the mid-80s to look into matters related to emergency preparedness and ground control in Ontario's underground mines, a renewed sense of commitment was made to understand how ground control related problems in Ontario's mines could be better addressed and the risks mitigated. From that grew initiatives and technology developments which have helped to make the mining industry in this part of the world a global leader in safety awareness and in risk mitigation and management. A fundamental value of Glencore's Sudbury Integrated Nickel Operations is that our first priority in the workplace is to protect the health and well-being of all our people. We take a proactive approach to health and safety; our goal is continuous improvement in the prevention of occupational disease and injuries. Sudbury Integrated Nickel Operations is committed to ensuring the long-term viability and continued success of our business as well as supporting the sustainability of our local communities. CEMI's Smart Underground Monitoring and Integrated Technologies for deep mines (SUMIT) Program has helped us in this regard by not only addressing key technical and technology-related challenges concerned with ground stability but also by informing us of the energy reduction and usage optimization opportunities that exist – each a key area in which improvements will be essential if the underground mining industry is to be sustainable.

The leadership demonstrated by Laurentian University in facilitating, developing and guiding the coming together of SUMIT researchers from Ontario's foremost mining universities and others throughout Canada is to be commended as is the efficient and effective management of the SUMIT Program by CEMI – Centre for Excellence in Mining Innovation. My sincere congratulations and compliments are also extended to the mining companies which have participated so extensively. To the best of my knowledge, this degree of collaboration, involving so many universities as well as mining companies which allowed the utilization of their mines as "living laboratories" for the conduct of experiments is unique. Some 98 technical people were involved and upwards of 75 Highly Qualified Personnel were trained from some 960 hours of underground work conducted with no incidents or accidents to report. Many of these people have already gone on to find employment within the industry, are continuing their studies or are working within academia. One spin-off company was created and numerous patents filed. Further, the production of this publication will stand as a testament to the good work accomplished.

The mining industry needs R&D, is willing to participate in both its funding and execution as well as to implement its outputs. SUMIT demonstrates that clearly. Industry particularity appreciates the inter-University research approach facilitated by strong leadership at Laurentian University. It perhaps can be a model for future programs of its kind.

Peter Xavier *Vice President*

Sudbury Integrated Nickel Operations
A Glencore Company
CEMI Board Member

EXECUTIVE SUMMARY

In 2010, a submission for funding to the Ontario Research Fund was made by Laurentian University to undertake research related to smart Underground Monitoring and Integrated technologies for deep mines. The total program budget was $6.7M with eventual approved funding being split almost equally among ORF, Universities and, importantly, the private sector. The Principal Investigator at Laurentian was Dr. Peter Kaiser and management of the SUMIT Program was undertaken on behalf of Laurentian University by the Centre for Excellence in Mining Innovation (CEMI)in Sudbury under the direction of Damien Duff.

SUMIT grew from a series of workshops held with experts across the globe to discuss ways in which the issue of fault-slip-caused rock bursting in deep underground mines could be better understood and the associated risk mitigated. As well, related areas of study were suggested in the fields of energy management and communications and control technologies. These would be aimed at developing and advancing smart engineering techniques, technologies and tools to facilitate step-change advances in productivity, efficiency and energy optimization in underground mining at depth.

Significant collaborative input to SUMIT was provided by researchers at Queen's University and the University of Toronto, with contributions also from the University of Waterloo, the University of Alberta, the University of British Columbia and Carleton University.

Through research and development undertaken at these institutions with the support of Vale, Sudbury Integrated Nickel Operations – A Glencore Company, Rio Tinto, and the Ontario Government, SUMIT focused on three challenges associated with deep underground mining:

1. ROCK ENGINEERING

Rock mass characterization to "better see into the ground" and anticipate risks and interpret dynamic processes;

2. UNDERGROUND MINE CONSTRUCTION

Enhanced mine development for faster mine construction to increase economic returns; and

3. ENERGY OPTIMIZATION

Sustaining deep mines through energy optimization and underground environmental controls.

Some of the high-level achievements of this six-year collaborative program include the involvement of 98 highly skilled technical people made up of research leads, students, post doctorial fellows (PDFs) and other expert professionals from across Ontario and Canada. The SUMIT Program has aided in developing the next generation with a total of 75 HQP at the PDF level (7), PhD (19), MSc (30) and undergraduate levels (19). SUMIT has garnered academic recognition and continued to enhance knowledge transfer producing 69 peer-reviewed scientific journal articles (with 12 in the works) which have been cited 374 times in scientific and engineering literature. In addition, SUMIT has led to 83 conference presentations and the publication of 41 conference papers across the globe.

SUMIT succeeded in bringing the best of the best in mining research together to collaborate and produce real value under this initiative. Following all safety protocols, the researchers and their students executed several field programs at active mine sites (Vale's Coleman & Creighton; Newcrest's Cadia East, and Sudbury Integrated Nickel Operations- A Glencore Company's Nickel Rim South Mine), incurring 959 student hours of in-mine work safely at four test sites.

SUMIT has helped to spin off a new company, grow existing businesses, and to file patents for new products which are ready to commercialize. It has also developed new guidelines and software tools for mine designers and operators. It has enhanced the understanding of energy demand and usage on mine sites, thus making it more amenable to optimization. Finally, SUMIT has furthered our understanding of how new energy approaches may be adopted at mines to lower energy demand, limit greenhouse gas emissions and reduce environmental footprints.

NOTE: **An online version of this journal publication can be obtained at www.cemi.ca and www.cim.org**

Highlights of the Technical Accomplishments

Under the SUMIT Program significant accomplishments were achieved in the following key areas: rock engineering, underground mine construction, energy optimization, and data analysis and management. Some are listed below:

ROCK ENGINEERING

- New software tool facilitating the selection of optimal ground support strategic designs based upon existing or anticipated rock mass conditions including a new coupled numerical modeling technique developed to study ground motion distribution and seismic wave propagation around excavations in deep mines.
- New industry guidelines document for the selection of ground support elements in underground mines.
- Demonstration that borehole geophysical data provide reliable estimates of geotechnical parameters such as dynamic Young's modulus at elevated in situ stress levels for rock mass characterization.
- Fiber optics cable borehole installation methodologies and performance measurement studies demonstrated the capability of fiber-based deformation devices in tensional and shear displacement conditions and provide a very good assessment of the evolving displacement field around a mining block due to mining.
- Advancement in the state-of-art and state-of-practice techniques for using continuum numerical models to replicate rock mass deformation and a demonstration of their ability to replicate actual behaviours observed in-situ.
- Development of an approach to treating LiDAR data which optimizes its use for deformation change detection.
- Development of an approach to reducing geotechnical assessment uncertainty as a function of the uncertainty associated with the conceptual geological model.

UNDERGROUND MINE CONSTRUCTION

- A new approach developed to differentiate how rock mass strength can be modeled in low and high confinement conditions.
- An improved understanding of the shear rupture of brittle rock under both constant normal stress and normal stiffness boundary conditions using calibrated numerical simulations.

ENERGY REDUCTION/OPTIMIZATION

- New approaches and tools developed to optimize energy usage on mine sites and save money.
- Optimized cooling strategies developed for natural heat exchange areas using computational fluid dynamics modeling.
- Optimal mine site energy supply options investigated and presented.
- Hydraulic Air Compressor (HAC) technology demonstrated as an economic alternative way to create compressed air for mines.
- Related to the above work and developed in conjunction with advancements made there have been significant ones related to data collection and analysis/management.

DATA ANALYSIS/MANAGEMENT

- A first of its kind data collection/sharing/management platform created which forms the basis for optimized geohazard assessment in deep mines.

Summary of Other Accomplishments of the Program

Aside from the technical accomplishments of the Program, it is worth drawing attention to those achievements in other areas, most notably, in H&S; in collaboration demonstrated with the private sector (active test sites which provided access for researchers and students to take measurements and collect data), in numbers of HQP (Highly Qualified Personnel) trained; in numbers of publications and citations, the success of SUMIT funds leveraging programs and in various outreach activities as well as in commercialization outcomes.

As well, the clear focus on ensuring that industry's needs were addressed were achieved through the feedback received from six consultation workshops held during the program scoping stage. During program execution, annual sessions with stakeholders were held in Toronto, Kingston, Waterloo and, finally, Sudbury to provide an update on progress being made.

Additional program value was also achieved through the significant leveraging of program funds through NSERC as well as through the federal Business-Led Network of Centres of Excellence program (BL-NCE), which led to the establishment of the Ultra deep Mining Network (UDMN) at CEMI.

The BL-NCE funding has led to the establishment, in conjunction with SNOLAB and CMIC (the Canada Mining Innovation Council) of a data analytics incubation space in Sudbury. It is anticipated that in the coming years, this will establish this part of Ontario as a knowledge creation and commercialization hub for mining and exploration data analytics. As of writing, a start-up and SME are established at MODCC with a further three under consideration.

The UDMN is a $35M five-year program of commercialization-focused research comprised of four themes of R&D. Two of these: Theme 1 – Rock Stress Risk Reduction and Theme 2 – Energy Reduction are direct add-ons to work begun under SUMIT.

Finally, SUMIT research has led to the filing of 11 patents and the spinoff of one new company (Electrale Innovation Limited). Further, it has helped to establish Ontario as an international centre of knowledge relating to mitigating geotechnical and energy use risk in mining.

Health and Safety Excellence and Exemplary Industry Collaboration Demonstrated

The two go hand in hand in the SUMIT Program given the incredible degree of collaboration required among researchers and students alike and the test site owners, particularly at Sudbury Integrated Nickel Operations – A Glencore Company, and Vale in Sudbury. Significant collaborative input was also provided by Glencore's Kidd Operations, Kidd Creek operations as well as by Newcrest Mining's Cadia East Division in NSW, Australia.

Living Laboratories

Sudbury Integrated Nickel Operations, A Glencore Company and Vale in Sudbury allowed their Nickel Rim South mine and Coleman and Creighton mines, respectively, to become "living laboratories" permitting the performance of underground data collection programs through sampling and surveying by SUMIT researchers and their students.

In fact, a total of close to 1,000 hours of safe underground work was completed by researchers at these properties (Figure 1) and would not have been possible were it not for the incredible efforts of site personnel.

Whereas most underground time was spent by University of Toronto researchers at Sudbury Integrated Nickel Operations – A Glencore Company's Nickel Rim South (NRS) mine, followed by Queen's researchers at Vale's Creighton mine in Sudbury, additional work was performed at NRS mine by researchers from the University of Alberta. Dr. Benoît Valley, a rock engineer and researcher based at MIRARCO in Sudbury at the time, also conducted underground data collection at Newcrest's Cadia East Mine in New South Wales, Australia.

Highly Qualified Personnel (HQP)

Ninety-eight persons, including researchers (PI and Co-PI's), students, research engineers and the Program Manager (Damien Duff) were involved with the SUMIT Program (Figure 2). Most had completed their degrees by the end of June, 2016. A few, however, due to a late start-up, had not and will be completed by June 30th, 2017, the extended deadline granted by the Ontario Research Fund.

Many of the student HQP have gone on to fill academic and industry roles of importance including at the Colorado School of Mines and the University of New Brunswick as well as with various consulting and other firms, including Hatch Engineering, GeoControl Chile, GeoFirma Ottawa, Abitibi Geophysics, MDEng, Kingston, Jacobs Associates, Vancouver, BC.

Publications

Because of the strong student-training element of the SUMIT Program it is not surprising that many publications, conference presentations, posters and citations in the broader literature should result – a total of over 200, in fact (Figure 3).

The work reported on has been highly regarded in the public literature apparently, having been cited by other authors in other published material some 374 times. Selected references and pub-

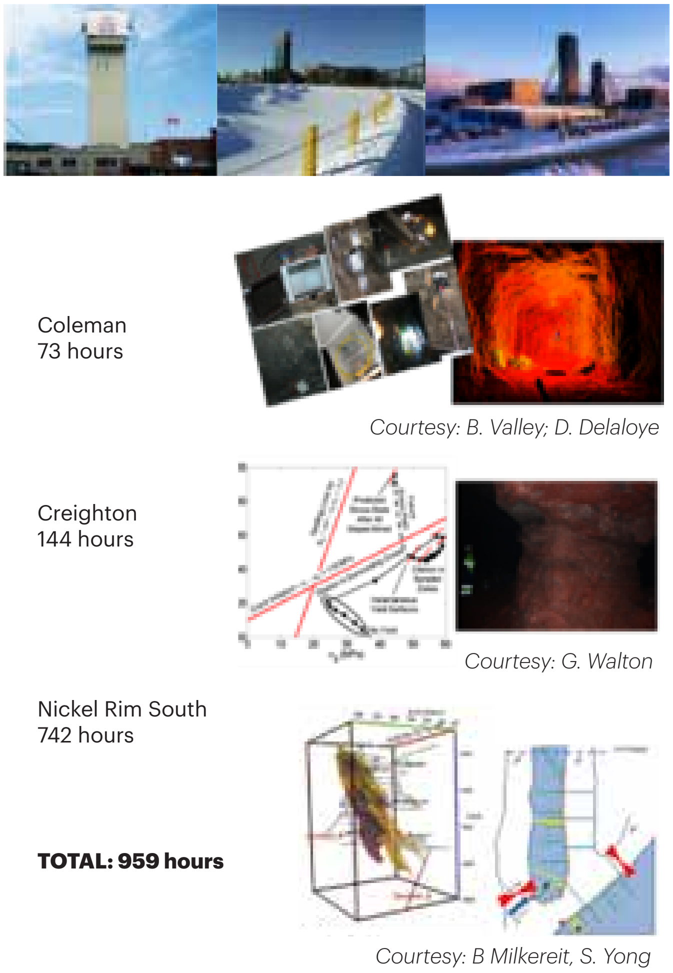

Figure 1: Location of underground fieldwork and hours completed

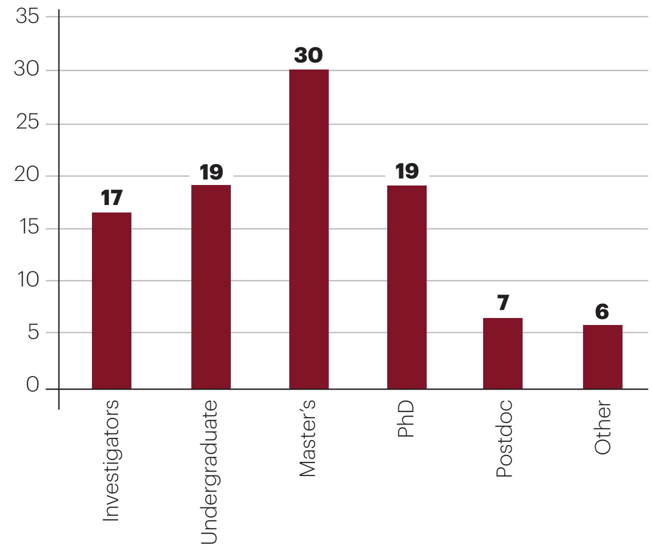

Figure 2: Breakdown by HQP Category, SUMIT Program

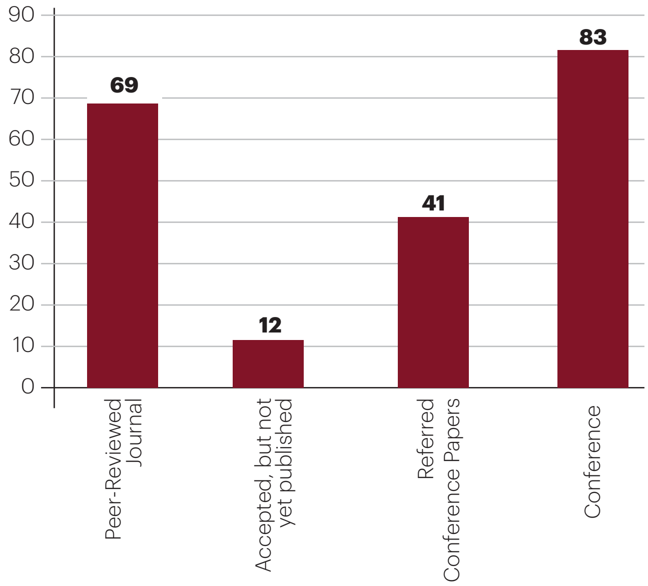

Figure 3: Publications and Presentations resulting from the SUMIT Program

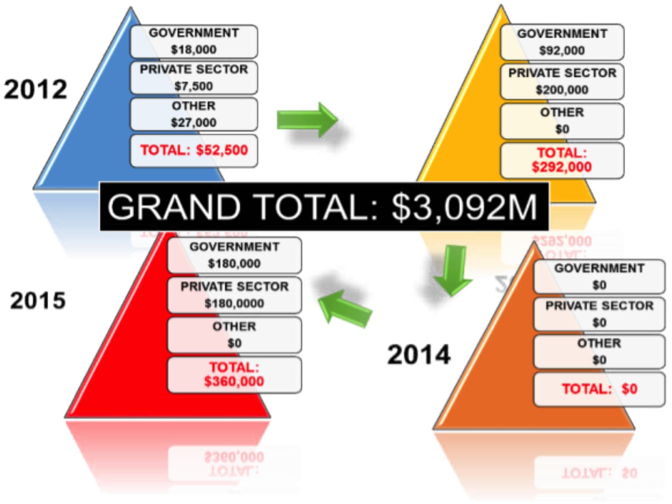

Figure 4: Leveraging performance of initial SUMIT Funding

lications are listed at the end of each of the technical sections of this document.

SUMIT Funds Leveraging

A measure of the success of some of the work conducted was always to be the degree to which researchers were willing to and able to leverage SUMIT investments in them through other government (Federal mostly – NSERC and BL-NCE) as well as industry funding sources. Starting in 2012, and continuing to 2015, with a hiatus in 2014 and most of the research already finished by 2016, the SUMIT investments were leveraged by a further $704,500 (Figure 4). An exceptional leveraging effort in 2016 brought this total to over $3M.

Participants at two SUMIT Annual Stakeholder Meetings held at Queen's University in Kingston (top) and at the University of Waterloo (bottom).

Commercialization

While never a strong focus for institutional researchers in general, some SUMIT Program researchers were successful in achieving some promising outcomes which have commercialization potential. These have been accomplished through the development of best practice guidelines, software tools and technologies or technology improvements. In one case, a new start-up company was spawned (Electrale Innovation Limited) and two new patents were filed (Laurentian University' s Dr. Dean Millar).

Outreach and Communications

Throughout the Program, communications with stakeholders was considered vital. This not only was to ensure continued support since it most definitely was needed, particularly from the test site owners, but also to ensure that all stakeholders felt they were being listened to and their work was being communicated to sponsors on a regular basis.

In addition to the Final Annual Stakeholders' meeting, held in Sudbury in 2016, other meetings and outreach events are summarized in Table 1

Additionally, regular communications were maintained (generally on a quarterly basis) with stakeholders via newsletters (Appendix: 3).

Leveraged Research

Ultra Deep Mining Network (UDMN):

Under Theme 1 – Rock Stress Risk Reduction of the Ultra Deep Mining Network program (UDMN) at CEMI, some $6M of work in seven projects is underway or is planned. It focuses on addressing stress measurement, geohazard assessment and rockmass behaviour management challenges in ultra-deep mines. Work in the latter area is a direct outgrowth of advancements in the field of hydraulic preconditioning of rockmasses begun under SUMIT.

Mining Observatory Data Control Centre (MODCC)

CEMI, in conjunction with the Canada Mining Innovation Council (CMIC); SNOLAB and the Northern Ontario Heritage Fund Corporation (NOHFC), has created a fully equipped Mining Observatory Data Control Centre (MODCC) within the world-class SNOLAB surface facility in Sudbury, Ontario, Canada dedicated to facilitating enhanced mining and exploration data analytics and management. The MODCC will be used to collect, store, analyze and share exploration and mining data sets. MODCC is a direct result of foundational work on data collection, management, and analysis, conducted as part of the SUMIT Program. See p.10-11 for more information.

Outreach Activity	Participants	Year Held
Seminars	72	2012
Student Presentations	27	2012
Short Courses	34	2012
Annual Workshops	113	2012 – 2016
Student Videos	8	2015
PDAC Student Workshop	27	2014

Table 1: Outreach Activities, year held and attendance

Acknowledgements

The huge effort required by Laurentian University in pulling this initiative together (largely through the efforts of Dr. Peter Kaiser as PI and his support staff) is acknowledged. As well, the substantial cash and in kind contributions of the principal industry partners in SUMIT: Sudbury Integrated Nickel Operations – A Glencore Company, Vale, Rio Tinto, were instrumental in enabling the leverage of institutional and provincial government funds for this program. Further, mine site personnel, designated as points of contact for the execution of planned SUMIT field work, are gratefully recognized. Their names are listed below:

Sudbury Integrated Nickel Operations, A Glencore Company –Mr. Brad Simser, Mr. Steve Falconer, Vale – Dr. Mike Yao, Mr. Allan Punkkinen, Stantec – Dr. Denis Thibodeau, (formerly at Vale); Dr. Fred Delabbio, (formerly at Rio Tinto).

Newcrest Mining Limited, also a collaborating Industry Partner, was represented by: Dr. Geoff Capes, (currently BHP Billiton Potash and Petroleum) and Mr. Rob Lowther.

The principal Government Sponsors were: the Ontario Research Fund (then Ministry of Economic Development and Innovation, now Ministry of Research, Innovation, and Science) and the National Science and Engineering Research Council (NSERC).

Myriad committees were necessary at the outset to ensure the proper scoping of SUMIT work, the organization of stakeholders and execution of plans and for this they deserve special mention. They include representatives at the VP Research level from participating institutions as well as at the VP or General Manager level from participating mining companies. Without their commitment to working together and facilitating the participation of others within their respective organizations SUMIT could not have happened.

At the political level, the support of the then Minister of Northern Development and Mines for Ontario, the Rt. Honourable Rick Bartolucci, was critical to ensuring connections to appropriate government departments and advice. His personal enthusiasm for all things mining-related was also greatly inspiring.

At CEMI, the efforts of virtually the entire organization were needed to have this R&D program happen at all and then eventually to manage it throughout its execution. All of them are hereby gratefully acknowledged.

Finally, significant technical expertise and assistance to CEMI was provided through Drs. Benoît Valley and Dr. Salina Yong, both at the time with MIRARCO in Sudbury.

APPENDIX

Program Overview/Technical Context and Delivery

In 2010, a detailed collaborative research proposal was submitted for Research Excellence Funding through the Ontario Research Fund by the Principal Investigator, Dr. P. K. Kaiser, at Laurentian University, with ten leading researchers from the University of Toronto, University of Waterloo and Queen's University. In 2011, the $6.7M program was approved by ORF, contributing $3. 3M to match industrial and institutional contributions.

The overall Research and Development (R&D) program aimed at ensuring that mines can safely and profitably mine at greater depth thereby securing the sustainability and competitiveness of the mining industry in Ontario. The SUMIT theme – Smart Underground Monitoring and Integrated Technologies – formed the framework for this research to build on previous experiences showing that the introduction of new technologies and smart monitoring approaches lead to major innovations. For example, the introduction of microseismic systems in underground mine construction some 30 years ago has led to much safer more economic mines at depth (fewer failures). Thanks to such innovations, the mining industry in Canada has become a world leader in exploration, mine design and mine safety.

The research program and related outcomes are illustrated in Figure 5 with a central pyramid of three focus areas:
- ROCK ENGINEERING
- UNDERGROUND CONSTRUCTION
- ENERGY OPTIMIZATION

SUMIT was designed to overcome challenges with advances in each of these three areas so as to:
- "see into the ground" and better anticipate and minimize risks
- enable faster mine construction to increase economic returns and to
- sustain deep mines through energy optimization with underground environmental controls

Ontario mining industrial partners committed funding and provided access to some of the world's deepest mines to serve as "living laboratories" to conduct closely linked experiments, each focusing on an enabling technology or technique, aimed at closing gaps in our ability to manage and control the underground mining process.

Thie SUMIT Program addresses several guiding principles of the Northern Ontario Growth Plan aimed at nurturing a knowledge-based economy, developing highly qualified personnel and helping to sustain mining communities.

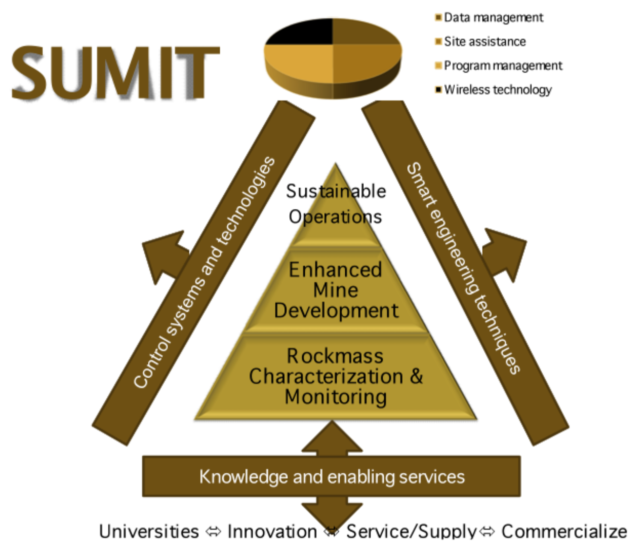

Figure 5

Technical Context

The SUMIT Program was developed to assist the Canadian & global mining sector in finding innovative means to reduce risk, accelerate mine development, improve production capacity and reduce energy consumption; specifically, to advance smart engineering techniques, technologies and tools to facilitate step-change advances in productivity, efficiency and effectiveness of underground mining at depth in an economic and safe manner.

It followed on from a total of six consultation workshops organized by CEMI involving international expertise from around the globe. Consultation sessions focused on various technical elements believed to contribute to the problem of anticipating rockmass behaviour because of mining. They included sessions on: microseismicity, structural geology, geophysics and instrumentation, modeling, planning and data integration as well as experimental design. A final close-out workshop pulled all elements together into a coherent framework in advance of application writing.

Starting out with a strong focus on fault-slip-caused seismicity, the eventual application documentation was re-framed in accordance with the likelihood of being funded through an ORF- round which had no mining focus but instead with an ICT (Information and Communications Technology) one.

The re-constituted program application aimed at using data and advancing technologies to mitigate the risks associated with deep mining more effectively. Ground instability and poor rockmass conditions in deeper mines are more prevalent than at a shallower depth which makes addressing them through engineering adjust-

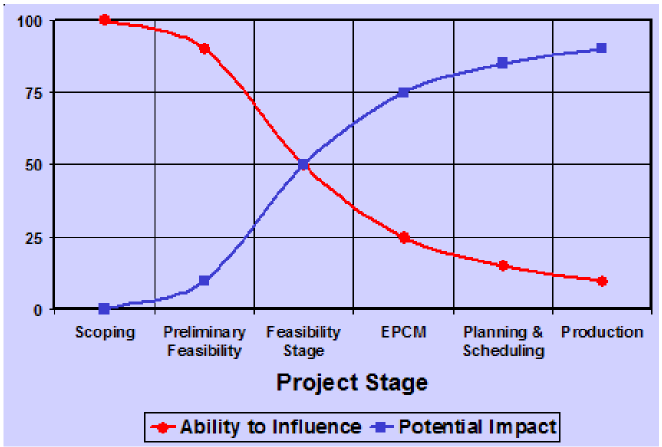

Figure 6: Trade-off between ability to influence and potential impact

ments much more difficult and costly. Therefore, it is essential to develop and operate deeper mines with the best information possible.

As with most major engineering projects, the ability to alter a major mine design decreases as it moves through the various engineering phases from scoping to production Figure 6). Hence, the potential impact of smart engineering is greatest in the early phases of a project where data and related information may be critically lacking. This is even more serious when dealing with costly and technically challenging deep mines.

After construction, and once mine operation has begun, the potential impact of step-changes in technology (e.g., mechanized excavation or rapid material handling) is similarly highest in the initial stages making early and successful adoption of such technologies vital. This is also important for the implementation of energy conservation measures, which are best done at the early stage of mine development (e.g. use of ventilation-on-demand at Sudbury Integrated Nickel Operations – A Glencore Company Sudbury's Nickel Rim South mine and Vale's Totten Mine).

With that, the data collection and management as well as the sustainability and energy optimization projects were added to the SUMIT Program scope of work.

Deep mines in Ontario (Sudbury) served as living laboratories for 18 core projects in 6 research categories (Figure 7 and Table 2).

With funding from Ontario's Ministry of Economic Development and Innovation and substantial contributions from Rio Tinto, Vale and Sudbury Integrated Nickel Operations – A Glencore Operation (formerly) Xstrata Nickel, SUMIT stimulated collaborative research among an outstanding group of researchers, consultants and industry experts.

The work at Laurentian University, the lead institution, was directed by Drs. Cai, Kaiser, Millar, and Smith; at Queen's University by Drs. Diederichs, Hutchinson and McKinnon (in collaboration with Dr. Samson at Carleton University); at the University of Toronto by Drs. Milkereit, Young and Xia (in collaboration with Dr. Schmitt at the University of Alberta) and, finally, at the University of Waterloo by Dr. Dusseault.

Significant consultant involvement occurred in the construction of a suitable data collection and management infrastructure for the SUMIT Program. This initiative was spearheaded by Mira Geoscience of Montreal, QB with significant early input by Objectivity Inc. of Sudbury, ON.

The program was managed and coordinated on behalf of Laurentian University by Damien Duff (VP Geoscience and Geotechnical R&D) at CEMI.

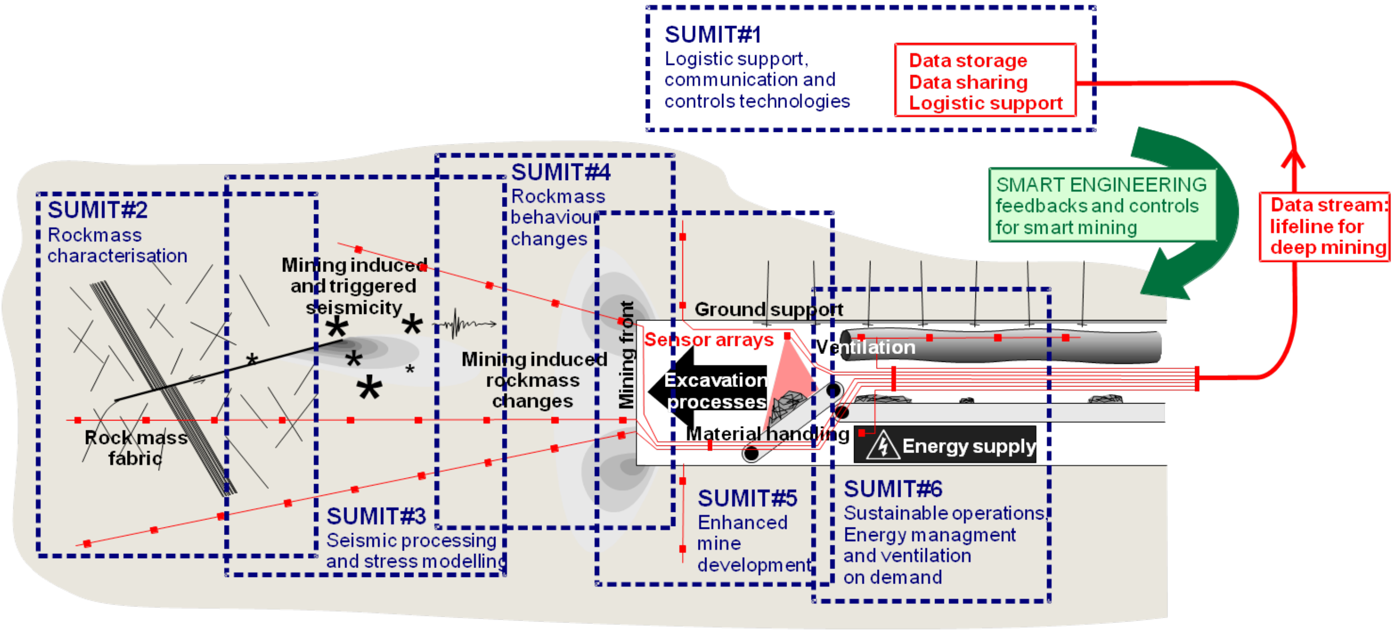

Figure 7: Research Categories within SUMIT

THEME	FOCUS OF WORK	RESEARCHERS
Shared Logistical Support And Information, Communications & Control Technologies	Ensure coordination of communications and site work with site owners. Develop sophisticated data analysis and management capability.	Damien Duff, Drs. Benoît Valley and Salina Yong at MIRARCO. Dr. John McGaughey, Mira Geoscience.
Rock Mass Characterization	Properly characterize the rock mass in advance of mining to ensure appropriate design and support selection. Monitor behaviour change duding mining to understand rock mass performance.	Drs. Mark Diederichs and Jean Hutchinson at Queen's University; Dr. Bernd Milkereit, Bibhuti Mohanti and Paul Young at University of Toronto. Dr. Doug Schmitt at University of Alberta.
Mining Induced and Triggered Seismicity	Integrate rockmass deformation and seismicity to correlate ground deformation response around openings to dynamic loading. Improve 3D remote-sensing monitoring technologies	Drs. Steve McKinnon at Queen's University and Claire Samson at Carleton University; Drs. Richard Smith and Peter Kaiser at Laurentian University. Dr's Bernd Milkereit, Qinya Liu at University of Toronto, Dr. Doug Schmitt at University of Alberta.
Rockmass behaviour and Mining-Induced Changes	Verification of constitutive model based on GSI for confined rock masses at depth. Assess rock mass failure of pillars and excavations	Dr. Peter Kaiser at Laurentian University; Dr. Mark Diederichs at Queen's University and Dr. Maurice Dusseault at University of Waterloo.
Enhanced Mine Development (Underground Construction and Design)	Identify key factors affecting ground motion in complex rocks within deep mines Provide methodology for ppv and ppa determination for improved rock support design. Develop and optimize fiber optics-based deformation measurement systems. Examine hydraulic fracturing mechanisms to mitigate energy release. Support introduction of mechanized excavation technologies by addressing constructability and pillar stability issues.	Drs. Peter Kaiser and Ming Cai at Laurentian University; Drs. Kaiwen Xia and Xija Gu at University of Toronto and Dr. Maurice Dusseault at University of Waterloo. Dr. Erik Eberhardt at University of British Columbia.
Sustainable Operations – Energy Management and Ventilation	Establish an understanding of energy economics as applied to the mining industry Develop tools and techniques to optimize energy usage at mines.	Dr. Dean Millar at Laurentian University

Table 2: Main SUMIT Themes, their technical focus and researchers

TECHNICAL SUMMARIES BY THEME

SUMIT #1a & 1b

#1a
Shared R&D Field Support

#1b
Support for Information and Communications Technology

MODCC
A Mining Observatory Data Control Centre for Northern Ontario

Shared R&D Field Support

The two aspects of this theme, field logistics support and ICT support were dealt with by separate teams. They are dealt with here in that order:

Team:

CEMI: working with MIRARCO – *Damien Duff*, CEMI Program Manager, SUMIT
Dr. Benoît Valley, Researcher, University of Neuchatel; *Dr. Salina Yong*, Vancouver, BC

Title: Damien Duff, SUMIT Program Manager
Role: Program Manager and Co-PI
Collaborators on team: Objectivity, Mira Geoscience, Sudbury Integrated Nickel Operations – A Glencore Company, Vale, Newcrest Mining Limited, MIRARCO, Laurentian University

Project Goals[2]

To provide all SUMIT researchers with support to facilitate mine site approval and access.

Abstract

Field logistical assistance and, where required, data collection and management coordination was provided by CEMI to all research sub-projects throughout the course of the SUMIT Program. Through contract engineering staff provided by MIRARCO at Laurentian University, Drs. Benoît Valley, followed by Salina Yong, coordinated all interactions with mine owner personnel at test sites. As referenced earlier, this totaled some 960 hours of accident and incident-free fieldwork.

Context

To address SUMIT Program R&D aspirations, extraordinary access to active mine sites (Living laboratories) for measurement and data collection would be essential. Because active mine sites offer the opportunity to observe and measure in the context of the changing real-world conditions accompanying the mining process, legitimacy, and direct application to the results from the analysis and interpretation of collected datasets would be more likely. The mines, in effect, would become "living laboratories".

Vale's Coleman and Creighton mines as well as Sudbury Integrated Nickel Operations – A Glencore Company's Nickel Rim South mine in Sudbury were experiencing the rock conditions and energy management challenges which made them highly suited to field work for SUMIT purposes. To a lesser extent other mines such as Glencore's Kidd Creek mine, Newcrest's Cadia East mine in NSW Australia and some of Codelco's operations as well as those at Antofogasta Minerias in Chile also offered opportunities.

Methodology

Post doc research engineers were hired with a focus on facilitating the interaction among researchers, their students, and mine site personnel to ensure consistency, a single point of contact and a minimum degree of mine site production disruption. Importantly, safety training requirements also needed to be met and arranged for.

Summary of Findings

Some 960 hours of underground work was conducted safely and successfully.

Practical Implications

With proper coordination and due regard to the production priorities of active mine sites, the data collection activities of large groups of researchers can be coordinated safely and effectively to yield valuable industry-focused results.

Conclusions

Highly effective underground research and development collaboration is possible between mining companies and academic institutions, even for large teams paying frequent visits to producing mines. This is made possible through the careful, single point contact method of communicating and coordinating activities.

Acknowledgements

Vale in Sudbury – Dr. Mike Yao, Allan Punkkinen, Anneta Forsythe, Glenn McDowell and Todd Madill are recognized. At Glencore's Sudbury and Timmins operations, Brad Simser, Steve Falconer and Dave Counter also require special mention. Others, more peripherally involved and perhaps at other mine sites but whose names are not known, are likewise thanked.

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[2] **Note: the nature of research is such that unforeseen circumstances (non-availability of students or suitably qualified ones, lack of access to appropriate test sites or non-availability of appropriate data) often dictate the ultimate details of the actual work carried out. Therefore, the research goals listed are only those which, given these constraints, received attention and for which results can be reported*

Support for Information and Communications Technology

Team:
Mira Geoscience – *Dr. John McGaughey* and *Mira Geoscience* staff Research Engineers and Geoscientists;
Andrew Dasys, Objectivity

Title: Dr. John McGaughey, CEO and CTO, Mira Geoscience
Role: Co-PI responsible for development of data management tool for SUMIT Program
Collaborators on team: Objectivity, Sudbury Integrated Nickel Operations – A Glencore Company, Vale, Rio Tinto

Project Goals

To centralize SUMIT data storage and facilitate seamless and prompt interaction among researchers on matters related to data so as to try to ensure quicker research outcomes, Mira Geoscience built a data management framework designed to match mine activity data (drilling, blasting, crushing, moving etc.– and thus signal producing) with sensor data. Further, established access protocols enabled uploading and downloading of datasets. Matching of sensor and activity data was accomplished partially but overall results suffered from difficulties in accessing activity data at the test sites.

Abstract

Geoscience INTEGRATOR is a 4D data management system for mining and exploration, developed by Mira Geoscience under the “Smart Underground Monitoring and Integrated Technologies” (SUMIT) program and the subsequent CEMI project, “Mining Observatory Data Control Centre” (MODCC).

The system design objective under SUMIT was a capability to “provide SUMIT researchers with user-friendly access to mine datasets and their contextual information to facilitate and optimize research efforts.” Under the MODCC project, exploration data themes and a powerful 3D visualization and query interface were added. It is currently being further developed in an Ultra Deep Mining Network (UDMN) project to support, track, and automatically report on 4D dynamic mine models and associated geohazards.

The system is now being deployed to manage multi-institution, multi-site data from significant research projects beyond SUMIT. This includes the largest mineral exploration research effort ever undertaken in Canada – the NSERC-CMIC Industrial Research Network project, “Integrated Multi-Parameter Footprints of Ore Systems: The Next Generation of Ore Deposit Models”. Geoscience INTEGRATOR will be released commercially in October 2016. It has both Canadian and international commercial customers committed to installations for managing data and geohazard assessments in underground mines. Although substantial further investment has been made in the system, the motivation, inspiration, and initial funding for this commercially successful outcome originate fully with the SUMIT Program.

Context

Systems for managing multi-disciplinary data are required for teams to share data, models, and interpretation for optimal geotechnical decision making. Microsoft Office products are used ubiquitously in the mining industry to manage and communicate day-to-day operational information – Excel for manipulation of numbers and PowerPoint for communication of ideas and results. Neither of these products was created to manage data or interpretation and they are unsuitable in many ways.

Data formats and file storage systems using such tools are idiosyncratic, meant to be understood by the individual data custodian rather than colleagues or the organization. Although drill hole, production, and microseismic databases are commonplace, they generally exist as silos, separate from each other and from other key data repositories such as geotechnical instrumentation or mine drawing files. Comprehensive systems and associated best practices to manage an organization’s geotechnical project data, interpretation, and decisions are not commonplace. The result is that interpretations that require or should be validated by multiple data types are exceedingly difficult to carry out in practice. Not only must primary data be located and exported from multiple systems, but contextual metadata is typically ad hoc and incomplete.

The need for a system to support, record, and communicate multi-disciplinary geotechnical interpretation is pressing as mines become deeper, geotechnical problems become more acute, and the flow of data available for analysis has ever greater speed and volume. Geotechnical hazard assessment, such as rockburst risk evaluation and monitoring, cannot be adequately addressed without access to a multi-disciplinary 4D data management system. This is because geotechnical hazard evolves in time, the underlying data are time-dependent, and the results of analysis must be routinely updated.

The SUMIT research program anticipated the need for a data management system in which researchers could solve complex problems that required collaborative analysis and interpretation of multi-disciplinary 4D underground mine data and associated contextual metadata. The objective of this component of the program was creation of such a system.

We have carried out many geotechnical hazard assessment projects for different classes of geohazard, at a wide range of mine types. We use a model-based approach in which individual criteria related to the hazard in question are captured in a spatial mine model and used as variables to be combined in a quantitative hazard assessment. Modelling the hazard criteria can be challenging, not least because many of the key criteria, and thus the resulting models, are four-dimensional. In practice the 4D nature of the problem is handled by time-stepping a 3D model. The modelling requires applying methods that map how data such as structure, rock quality, and seismicity, often located far into the rock mass away from development, manifest hazard within the mine development.

An important lesson from these studies is that operations typically do not have the data management infrastructure to readily provide data to a comprehensive, multi-disciplinary 4D analysis. This situation is manageable where hazard assessment updates are only required off-line on an intermittent basis. Data compilation is always the most expensive and time-consuming part of this type of project, particularly when there is a lot of multi-disciplinary data and several custodians are involved. It is our experience that if projects falter, they typically falter at this stage, which is a symptom of inadequate data management practices. Ongoing or real-time operational assessment of hazard requires up-to-date access to all the data types required for calculation, including microseismic and other monitoring data.

The SUMIT research program foresaw similar data management challenges, with its multiple case study sites, widely differing experiment types, and multiple research institutions and collaborators. A need was cited, according to the original scoping documents, for “creating an appropriate structure for collecting, standardizing, storing, and sharing mining data sets with research and development entities in a fully controlled manner.” This research need precisely mirrors the wider industry need for such a system as the foundation of strategies for interpretation and operational decision making when confronted with complex phenomena such as rockbursts in underground mines.

Requirements of the SUMIT data management system were outlined in the original scoping document as follows:

An important part of this component is to provide SUMIT researchers with user-friendly access to mine data sets and their contextual information in order to facilitate and optimize research efforts. The data and contextual information can vary from researcher to researcher. Equally important, the solution platform must offer a single point of access that minimizes demand on mining personnel by eliminating duplicate data requests.

The data sets that are needed for this project are created and stored by mining companies in disparate data management systems. An important component of the project is to put in place the protocols, adapters, and converters that bring these datasets in the data warehouse in consistent formats and to update them on a regular basis.

The following tools are envisaged:
– Topical data browsing
– Data querying/filtering
– Visual data browsing and querying
– Downloading

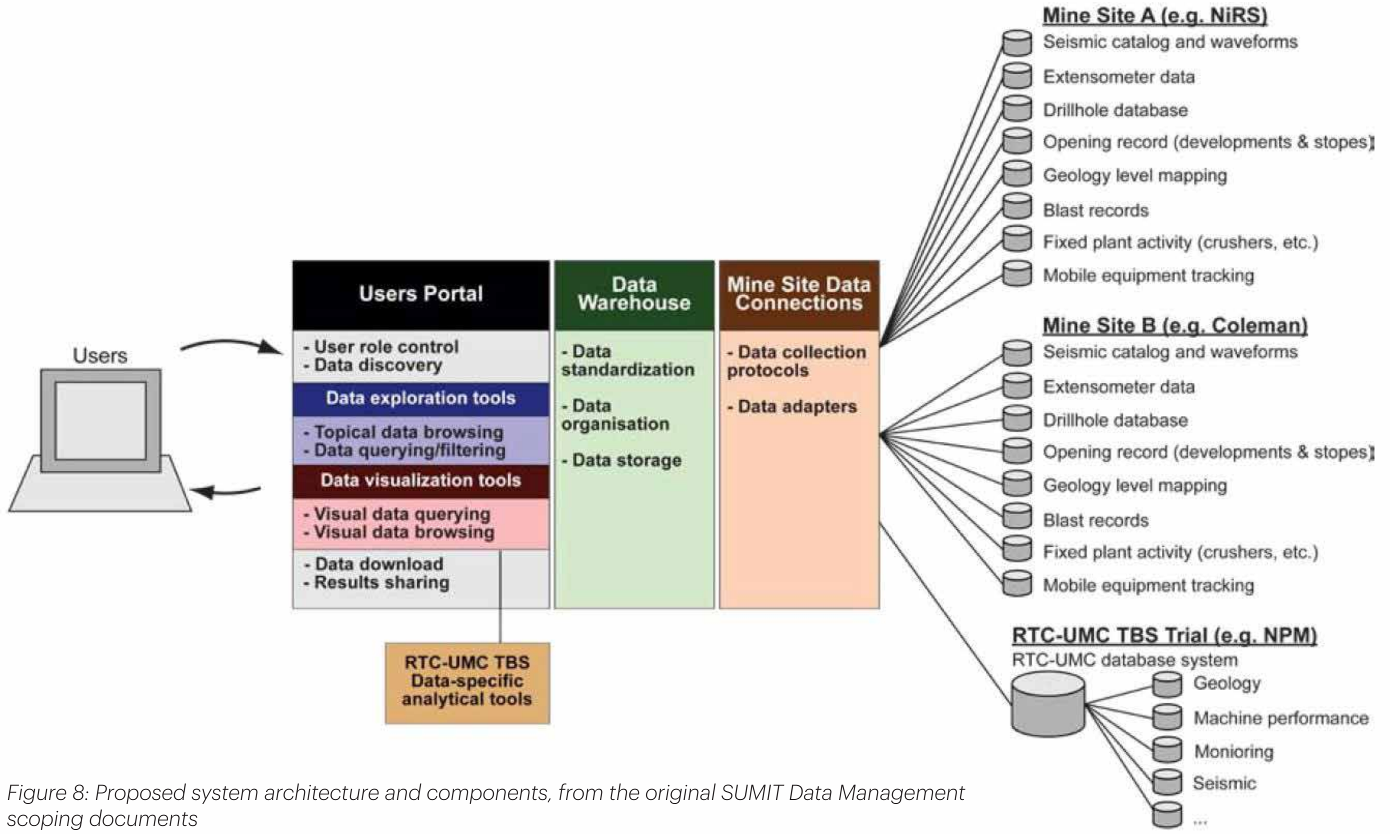


Figure 8: Proposed system architecture and components, from the original SUMIT Data Management scoping documents

- Posting back research results

For the SUMIT project, the following mine datasets are envisaged:
- Mine infrastructure, i.e. mine openings
- Seismic events catalog
- Seismic triggered waveforms
- Geology data (mapped and interpreted)
- Drill hole logging data
- Plant activity
- Mobile equipment activity
- Extensometer measurements
- Blast reports

A graphical summary of the required system from the original scoping documents is shown in Figure 8.

Methodology

The following design requirements to meet the system objectives were defined:
- Data is organized by mine site (or "project") and data type
- A permissions system defines which users have access rights to read, write, view, and download data
- All data have the capability of time-variance
- Data formats are user-defined
- A file and document management system enables linkages between data and associated files (such as image or other binary files) and documents
- A "tagging" system is used to label and query data
- The primary system interface is a web browser that can be used on a desktop or tablet device
- A powerful, desktop 3D visualization application is used for querying and viewing data in the 3D mine context
- A reporting system allows reports to be generated on user demand or according to a schedule, in which case the reports are automatically created and emailed to a specified user group.
- The system contains an Application Programming Interface (API) that allows third-party applications to communicate with it.

Summary of Findings

A system that meets the design requirements has been successfully developed and deployed. Figure 9 illustrates a basic system schematic, showing key components including the back-end database server, the mail server for report distribution, the web browser client as desktop and tablet application, the 3D visualization and query interface ("Geoscience ANALYST"), and the main Geoscience INTEGRATOR server at the centre of the system. A screen image from the Geoscience ANALYST interface is shown in Figure 10.

Practical Implications

An innovative and practical data collection, storage and retrieval system has been built and forms the basis for further developments which will help ensure maximum value is derived from mine datasets, including by using machine learning applications.

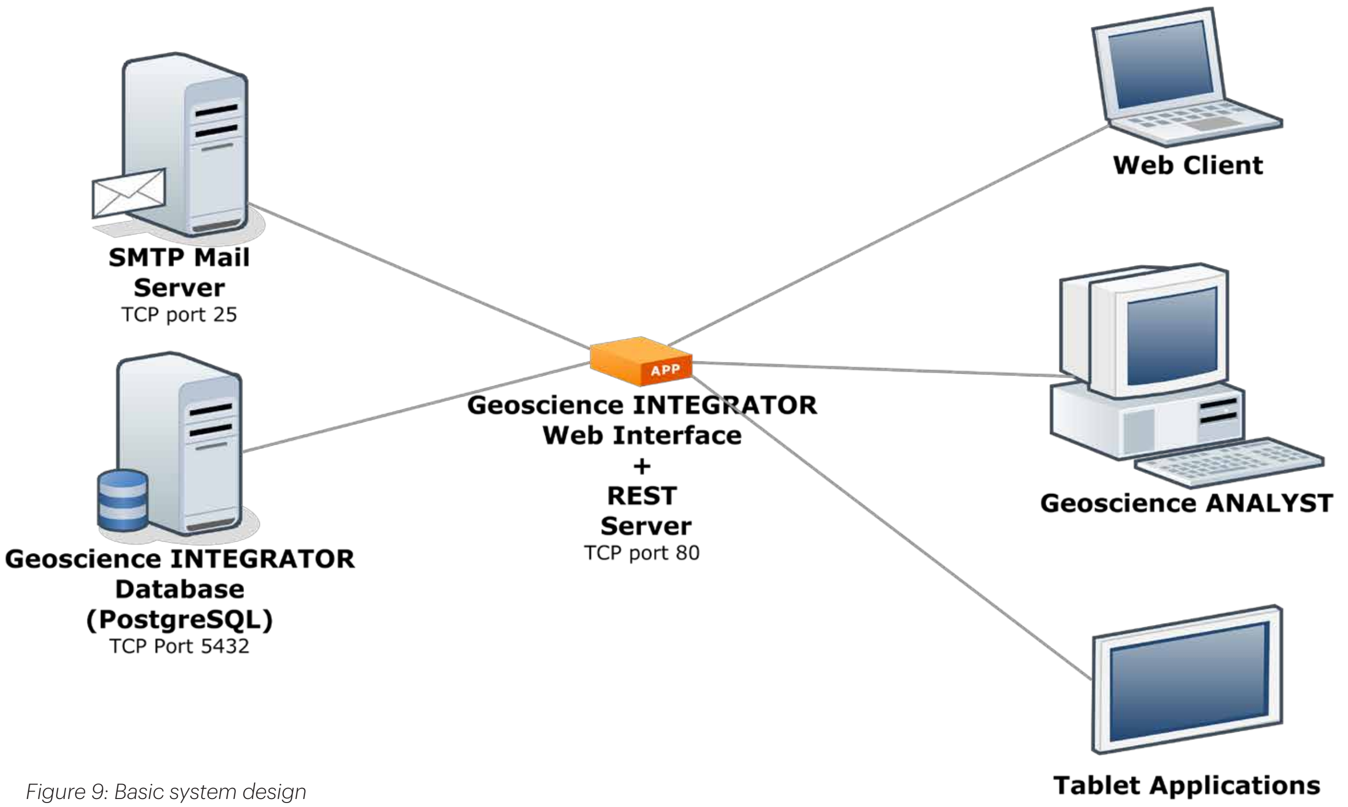
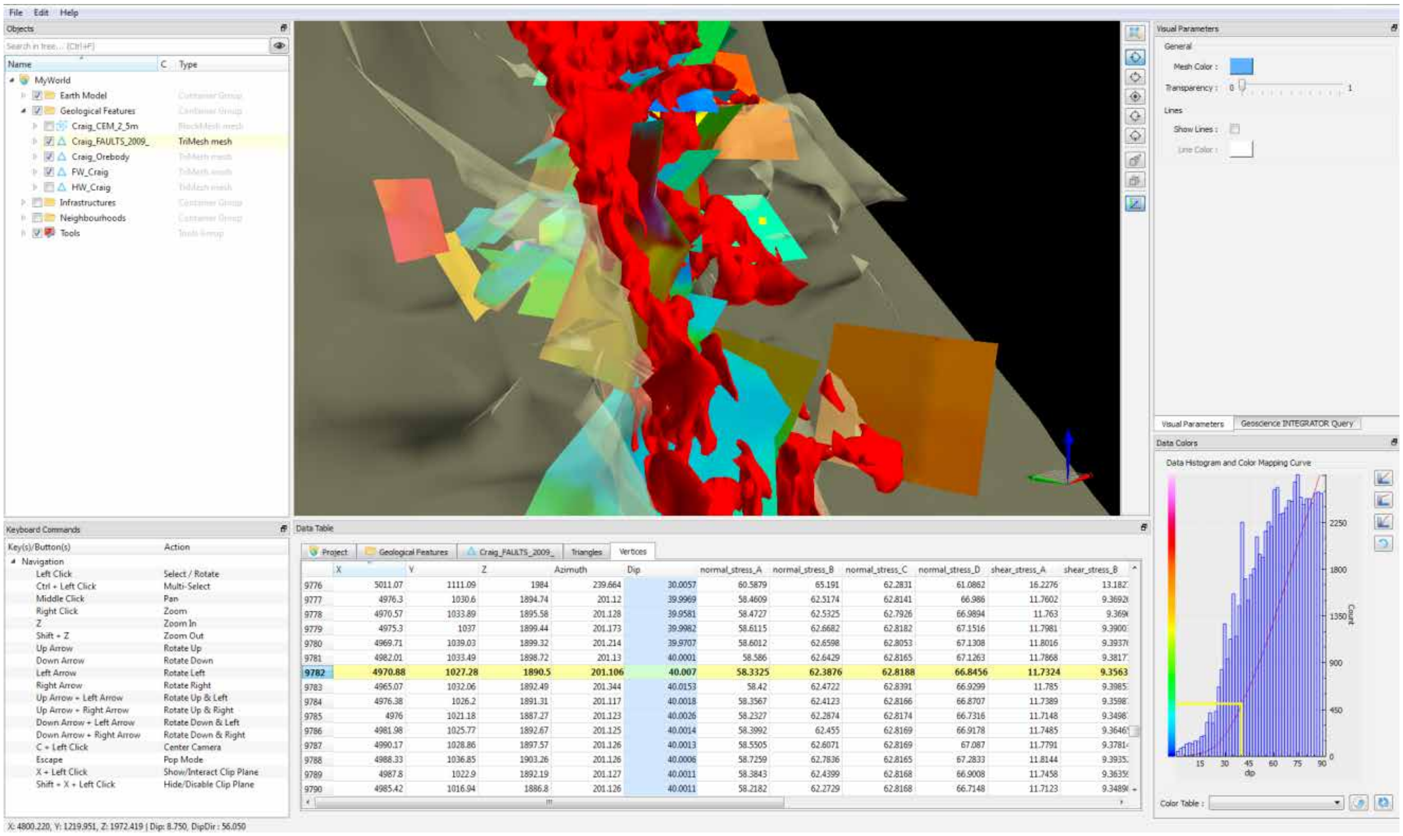


Figure 9: Basic system design

Conclusions

Although required development time meant that the system was not available at its full intended scope for most of the SUMIT project, its document management and time recording functionality was used by project participants. The system is being deployed as a SUMIT data archive in 2016. Through the CMIC "Footprints" project, new exploration-themed data types have since been developed. Also through the Footprints project, which continues to March 2018, the system is being extended to include machine learning functions which will enable insights to be developed from the wealth of data held within the system. Machine learning applications, deployed directly within the system, are being specifically researched for the geotechnical hazard assessment problem.



Key(s)/Button(s)	Action
Navigation	
Left Click	Select / Rotate
Ctrl + Left Click	Multi-Select
Middle Click	Pan
Right Click	Zoom
Z	Zoom In
Shift + Z	Zoom Out
Up Arrow	Rotate Up
Down Arrow	Rotate Down
Left Arrow	Rotate Left
Right Arrow	Rotate Right
Up Arrow + Left Arrow	Rotate Up & Left
Up Arrow + Right Arrow	Rotate Up & Right
Down Arrow + Left Arrow	Rotate Down & Left
Down Arrow + Right Arrow	Rotate Down & Right
C + Left Click	Center Camera
Escape	Pop Mode
X + Left Click	Show/Interact Clip Plane
Shift + X + Left Click	Hide/Dicable Clip Plane

	X	Y	Z	Azimuth	Dip	normal_stress_A	normal_stress_B	normal_stress_C	normal_stress_D	shear_stress_A	shear_stress_B
9776	5011.07	1111.09	1984	239.664	30.0057	60.5879	65.191	62.2831	61.0862	16.2276	13.182
9777	4976.3	1030.6	1894.74	201.12	39.9969	58.4609	62.5174	62.8141	66.986	11.7602	9.3692
9778	4970.57	1033.89	1895.58	201.128	39.9581	58.4727	62.5325	62.7926	66.9894	11.763	9.369
9779	4975.3	1037	1899.44	201.173	39.9982	58.6115	62.6682	62.8182	67.1516	11.7981	9.3900
9780	4969.71	1039.03	1899.32	201.214	39.9707	58.6012	62.6598	62.8053	67.1308	11.8016	9.3937
9781	4982.01	1033.49	1898.72	201.13	40.0001	58.586	62.6429	62.8165	67.1263	11.7868	9.3817
9782	**4970.88**	**1027.28**	**1890.5**	**201.106**	**40.007**	**58.3325**	**62.3876**	**62.8188**	**66.8456**	**11.7324**	**9.3563**
9783	4965.07	1032.06	1892.49	201.344	40.0153	58.42	62.4722	62.8391	66.9299	11.785	9.3985
9784	4976.38	1026.2	1891.31	201.117	40.0018	58.3567	62.4123	62.8166	66.8707	11.7389	9.3598
9785	4976	1021.18	1887.27	201.123	40.0026	58.2327	62.2874	62.8174	66.7316	11.7148	9.3498
9786	4981.98	1025.77	1892.67	201.125	40.0014	58.3992	62.455	62.8169	66.9178	11.7485	9.3646
9787	4990.17	1028.86	1897.57	201.126	40.0013	58.5505	62.6071	62.8169	67.087	11.7791	9.3781
9788	4988.33	1036.85	1903.26	201.126	40.0006	58.7259	62.7836	62.8165	67.2833	11.8144	9.3935
9789	4987.8	1022.9	1892.19	201.127	40.0011	58.3843	62.4399	62.8168	66.9008	11.7458	9.3635
9790	4985.42	1016.94	1886.8	201.126	40.0011	58.2182	62.2729	62.8168	66.7148	11.7123	9.3489

X: 4800.220, Y: 1219.951, Z: 1972.419 | Dip: 8.750, DipDir : 56.050

Figure 10: The Geoscience ANALYST desktop interface used for 3D visualization and query of the data management system.

MODCC – A Mining Observatory Data Control Centre for Northern Ontario

Team:

CEMI, SNOLAB, CMIC (Canada Mining Innovation Council), Ontario Government (NOHFC)

Title: Damien Duff, Program Manager; Dr. Nigel Smith, SNOLAB Director
Role: Coordinate the funding, construction, and occupation of MODCC
Collaborators on team: Ontario Government, CMIC, Objectivity, Mira Geoscience

Project Goals

Based upon the data collection, storage and sharing needs of the SUMIT Program and the technology developed within SUMIT to handle them, build a fully equipped facility which promotes the location and development of globally-engaged exploration and mining data analytics expertise in Northern Ontario.

Abstract

MODCC is a data analytics incubation space developed in a collaborative effort involving CEMI, SNOLAB and the Canada Mining Innovation Council (CMIC), with some funding provided through the Ontario Government (NOHFC).

It has been set up at the SNOLAB surface facility as both a physical and virtual location to foster interdisciplinary data analysis which promotes data-driven innovation for the exploration and mining industries.

Context

The scope of the SUMIT Program, involving as it did so many researchers at different institutions, coupled with a desire which CEMI had to stimulate a paradigm shift in collaboration and the speed of R&D outcomes to industry, necessitated a new and modern approach to data management.

Conventional R&D approaches involve the collection and subsequent analysis of data leading to conclusions released through a time-honoured journal publication process. Thus, it can take years for these data, dealt with in this manner, to become exposed. The rate of R&D outputs which, once released, may lead to significant improvements for industry, is thus slow. Modern data management, dissemination and visualization tools are needed to change this paradigm and bring positive research solutions to industry faster.

It was envisaged that modern digital tools, once in place, could be accessed and used by SUMIT researchers to safely store their SUMIT data while at the same time being able to share it among themselves (using strict protocols) and thereby benefit from the additional value each could bring.

Methodology

Once Geoscience INTEGRATOR was built and its capability and true potential to be a game-changer for managing large exploration and mining datasets became apparent, a mechanism whereby it could become foundational to the establishment of a new data-analysis and management-oriented centre based in Northern Ontario needed to be found.

In conjunction with SNOLAB and CMIC, CEMI (including cash and in kind contributions from each) applied for and received substantial funding from Ontario's Northern Ontario Heritage Fund Corporation (NOHFC) to build a centre located at the SNOLAB surface facility in Lively, Ontario. Upon funding and the completion of construction, a global Request for proposal (RFP)

was issued by CEMI to encourage applications for incubator positions in data analytics at MODCC.

At the same time, a formal review committee was established comprised of qualified individuals from Ontario Centres of Excellence (Ron van Holst); the DMZ at Ryerson University (Fil Varino); MaRS Discovery District (Joe Lee), SNOLAB (Chris Jillings) and CEMI (Brian Jones, Mike Richer and Damien Duff).

Summary of Findings

MODCC has been successfully built at the SNOLAB surface facility as both a physical and virtual location to foster interdisciplinary data analysis which promotes data-driven innovation for the exploration and mining industries.

Practical Implications

MODCC is an operating entity presently with two data analytics/management start-ups and one SME in residence for an initial 2-year incubation period. Each is working on algorithms and related software products with a plan to commercialize them at or near the end of their 2-year tenure at MODCC. Mira Geoscience plans to make MODCC a hub for its mining data analytics/management commercial service, beginning in Q1, 2017.

Conclusions

The time for novel data analytics approaches and their adoption by the mining industry has come. Northern Ontario can become a knowledge centre for data analytics for the mining industry.

Some facilities shared with SNOLAB

SUMIT #2

Rockmass Characterization for Deep Mines

2.1

Rockmass Behaviour Characterization and Optimal Use of Existing Data to Develop Representative Models

2.2

Rock Mass Strength at Depth

2.3

Automated Characterization and Change Detection Within 3D Digitally Scanned Virtual Excavation Boundary and Borehole Models to Quantify Rockmass Deformation Response to Mining Over Time

Rockmass Characterization for Deep Mines

Title: Dr. Peter Kaiser, Professor Emeritus
Role: Principal investigator responsible for delivery of SUMIT technical program
Collaborators on team: Rio Tinto, Vale, Sudbury Integrated Nickel Operations – A Glencore Company, Newcrest Mining Limited

Objectives

As mines progress to greater depths, the ground becomes less forgiving and careful ground characterisation practices must be followed to establish representative and reliable rock mass properties. It was hypothesized that the rock mass strength at depth is drastically underestimated when conventional characterization tools and techniques are applied. Hence, the primary goal of this theme is to develop reliable means for rock mass strength determination in green fields situations.

This involves amongst others:

- Improvements in synthesis of borehole data to refine relations between borehole investigation data and rockmass parameters for deep mining.
- Establishing limits of applicability of currently available semi-empirical techniques such as the Geological Strength Index (GSI) and developing new methods if not applicable.
- Refinement and validation of discrete fracture network generation tools based on borehole data.
- Characterizing the initial state and changes in the rock mass properties during mining using borehole petrophysical and borehole geophysical studies.
- Detection of rockmass changes using LiDAR or other surface imagery techniques and development of automated algorithms for change detection using dense point cloud data in boreholes or drifts (ATV data or LiDAR).
- Deformation measurement interpretation through tress-strain back analyses.
- Development of Georisk assessment procedures to guide systematic risk assessment for deep mines and an S-GMAT tool for ground motion simulation/monitoring.

CONTEXT FOR THEME 2

"Underground rock engineering to match the rock's behaviour – Challenges of managing highly stressed ground in civil and mining projects"

The context for Theme 2 is well described by Dr. P K Kaiser's 2016 MTS lecture and the key messages of the lecture are outlined in the ARMA and ISRM newsletters (Kaiser 2016) in the form of a brief executive summary with selected extracts from the slide deck. The reader is referred to related publications for detailed explanations.

In the spirit of the conference theme "new exciting advances in rock mechanics", this lecture introduced several recent advances that are awaiting application in engineering practice and aimed at opening new paths of discovery by questioning implicit assumptions in standard engineering approaches.

A robust rock engineering solution in underground mining or construction must respect the complexity and variability of the geology, consider the practicality and efficiency of construction and provide safe and effective rock support.

For this purpose, it is essential to anticipate the rock mass and excavation behaviour early in the design process. Whereas it is possible in most engineering disciplines to select the most appropriate material for a given engineering solution, in rock engineering, a design must be made to fit the rock, not vice versa.

Lessons learned from excavation failures (Figure 11) tell us that stressed rock at depth is less forgiving and that advances in rock mechanics demand a full comprehension of the behaviour of stress-damaged rock near excavations.

Comprehension in this context means explaining all observations such that fiction can be separated from reality and our engineering models and methods become congruent with the actual behaviour of a rock mass. The abridged version of the lecture (Kaiser 2016) presents a brief discussion of aspects of underground rock engineering where dichotomies exist and gaps between reality and current practices have to be closed by the application of recent advances in rock mechanics to arrive at sound rock engineering solutions.

Figure 11: Examples of excavation instabilities in stressed rock

Much progress has been made in recent years in rock mechanics and these findings are now available for better and more robust engineering designs, i.e., designs that match the rock mass and excavation behaviour and thus can be constructed without undue delays and without excessive costs.

Furthermore, observations of rock behaviour to verify the applicability of design approaches are essential for sound and robust engineering. With systematic observations and correct interpretations, fiction can be replaced by models of reality and solutions can be found that fit the rock.

Publications

Kaiser P.K 2016. Underground rock engineering to match the rock's behaviour – a fresh look at old problems. MTS lecture at 50th US Rock Mechanics Symposium, p.6.

Team:

Queen's University – *Drs. Mark S. Diederichs* and *Jean Hutchinson* with students: *Gabe Walton, Jenn Day, Connor Langford* and *Ioannis Vazaios, Cortney Palleske, Dani Delaloye, Felipe Duran, Michelle vdP Kraan* and *Shaun O'Connor.*

Title: Drs. Mark Diederichs and Jean Hutchinson
Role: Co-PI's
Collaborators on team: Vale, Codelco, Antofogasto Minerias, Sudbury Integrated Nickel Operations – A Glencore Company

Abstract

This work focussed on geological and geotechnical inputs into analysis and decision making for mine design, including support considerations. The Geotechnical Model is intimately dependant on the Geological Model, although modern rock engineering techniques often ignore this association. Within this component of the work, led by O'Connor (2015), the linkages between geological modelling decision making and the resultant magnitude, variance and importance of key geotechnical parameters is explored primarily with a view to highlighting this important linkage as critical to modern site investigation.

As well, this project has built on existing expertise in the use of laser scanning (LiDAR) and data processing to extract geotechnical data, progressing to the use of LiDAR data to accurately determine and map deformations and change within underground excavations. This represents a significant step forward in the use of field data for model validation and calibration, leading to increased reliability of design modelling.

Context

The work summarized here was all executed within a common framework of improving data collection tools for geotechnical input, improving geotechnical analysis tools for predicting rockmass movement and response, and improving monitoring and measurement tools for model inputs, validation and calibration. Mines are being developed in deeper environments, with increased upfront development and in increasingly complex geological and geomechanical domains. These challenges present an inevitable increase in geohazard potential for the early development and for the mining lifetime as a whole.

The key components of Geo-Risk can be summarized as shown in Figure 12.

In this flow chart, uncertainty is first introduced into the geotechnical design process through the formation of the geological model. While there is some feedback from logistical mine design and from geotechnical model generation, the majority of this geological model is formulated during the exploration and evaluation stage for an orebody. The geotechnical model is then normally "piggy-backed" onto the geological model. It is at this geotechni-

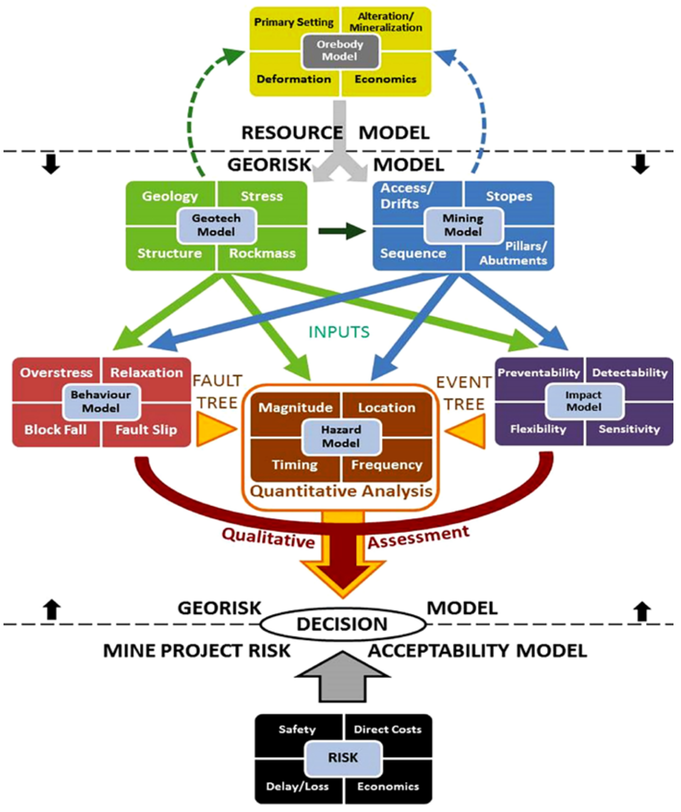


Figure 12: Geo-Risk framework for mine design

cal stage that most past work related to engineering uncertainty has been carried out, with the result that the variability and reliability of geotechnical parameters have been quantified to varying degrees. There is, however, an equal or greater uncertainty with respect to anticipated failure or behavioural modes.

Such analysis models only benefit from enhanced input data if there is confidence that the modes themselves are reliably predicted. For most projects, there will be uncertainty related to the mining model. Designs can be sensitive to initial planning and it is common that mine plans get modified on a strategic and tactical level throughout the mine life. It is important to consider this sensitivity. The impact model for rockmass behaviour, while understood, is often not formalized. It is important to understand the roles of monitoring (detection), operational sensitivity and flexibility, and preventive options in the overall project Geo-Risk. All of these aspects feed into qualitative or quantitative analyses of hazard within the project. The end result is therefore a measure of georisk that can feed into decision making based on a risk acceptability and tolerances within the project.

This model, developed by the Queen's Geomechanics Group with past CEMI support is an important framework from within which the contributions of the three recent Queen's Geological Engineering SUMIT subprojects can be conceptualized.

2.1 Rockmass Behaviour Characterization and Optimal Use of Existing Data to Develop Representative Models

Context

The Geotechnical Model is intimately dependant on the Geological Model, although modern rock engineering techniques often ignore this association.

Methodology

Early geological studies, structural and geotechnical investigations to identify and more rationally analyze rockmass behaviour for geotechnical mine design were integrated. Rockmass strength, structural control and secondary influences are the focus with an emphasis on improving reliability of measurement while considering the natural uncertainty (variability) within the associated geomaterials and complex geosystems associated with mining. Ultimately, this work feeds into analysis models where the uncertainty can be properly considered.

Summary of Findings

The linkages between geological modeling decisions and key geotechnical parameters are critical. Figure 13 illustrates this associated with a few examples. Based on the level of exploration data available, the number of geological components identified within local regions of the model and linkages to geotechnical parameters, the uncertainty of the geological interpretation can be assessed (dependent on data density but also on the number of competing and possible alternative interpretations) as can the relative sensitivity of geotechnical parameters to the small or large variants in geological interpretation. A schematic association is shown in Figure 14.

A second component of this sub-project involved testing and validating tools that are now available for the creation of virtual rockmasses, namely discrete fracture networks such as that gen-

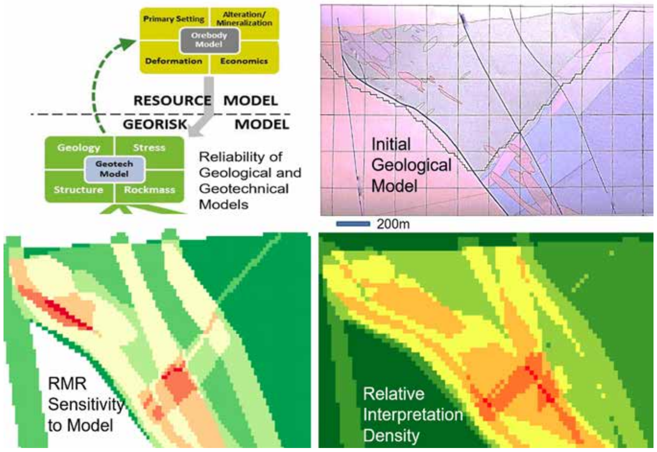


Figure 13: The geological model for an open pit and associated interpretation density (amount of geological input per volume) and geotechnical sensitivity (after O'Connor)

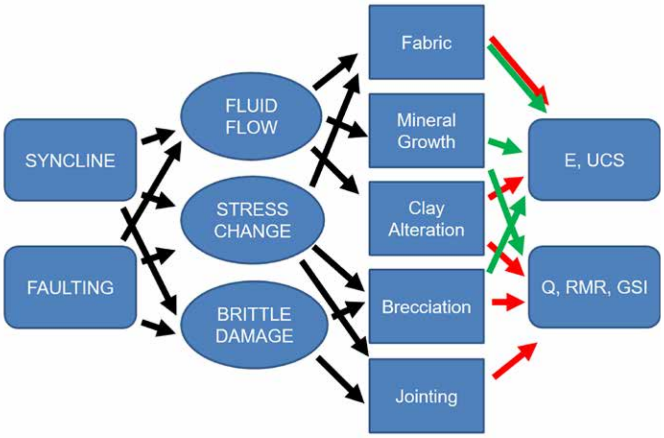


Figure 14: Geological factors impacting geotechnical parameters after O'Connor et al

erated from the photogrammetric scan in Figure 15. These new tools show promise to move rock engineering beyond the standard practice of rockmass classification coupled to empirical or continuum analytical/numerical analysis.

These DFN models are significantly complex, however, and validation and/or verification of the model is often impossible or impractical at a large scale. It is essential to understand the limitations and sensitivities of these DFN modelling tools and the resultant models to input variability, bias and sampling artifacts as illustrated in Figure 16.

This component of the sub-project, led by Palleske (et al) has been successful in identifying critical considerations for valid modelling of this type and providing guidance to the engineer or geologist user of these tools with respect to data

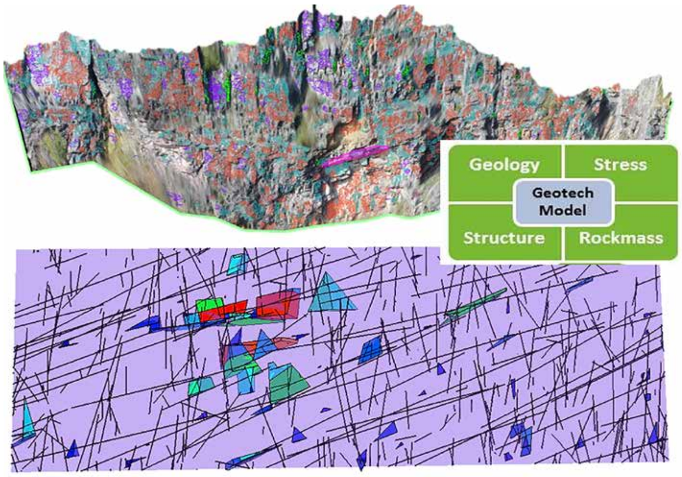


Figure 15: DFN (lower) generated from photogrammetry survey (upper) after Palleske et al

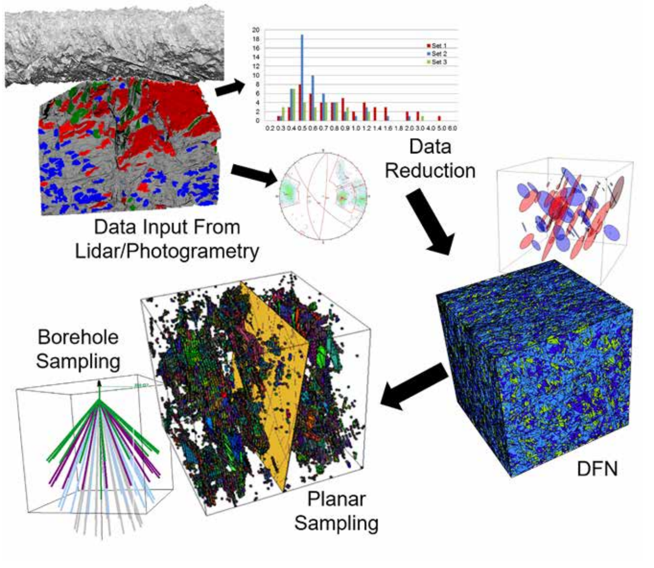


Figure 16: Workflow for processing LiDAR data to obtain structural data to generate a DFN which can be probed and sampled (after Palleske et al)

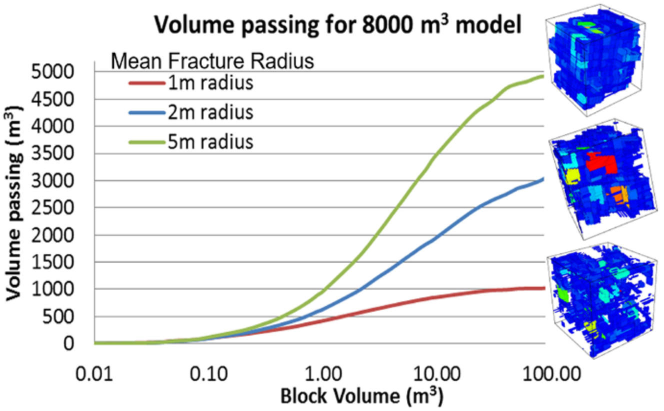


Figure 18: Example of DFN query for investigation and application (block size) after Palleske et al

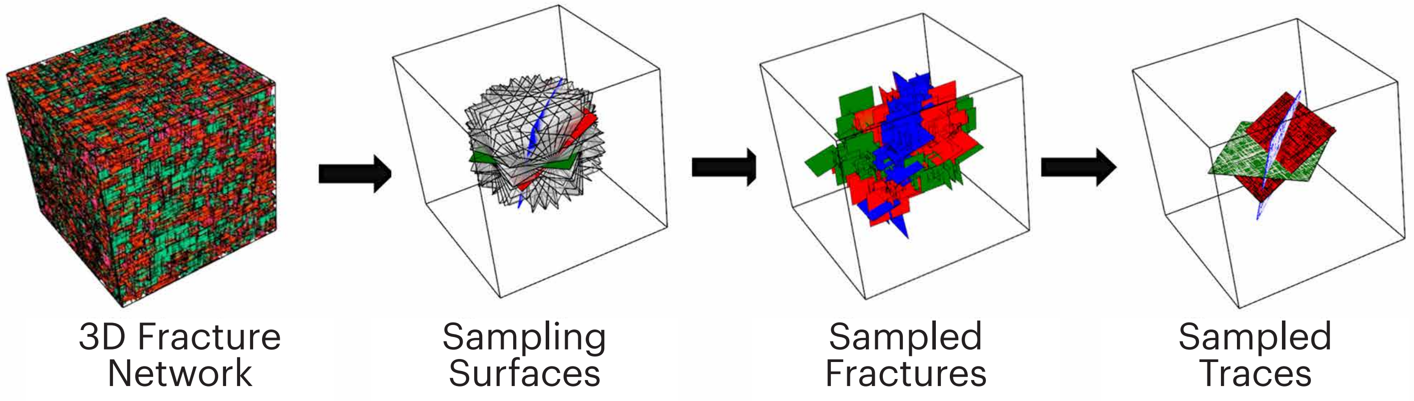


Figure 17: Workflow for probing a 3D DFN to obtain 2D calibration data

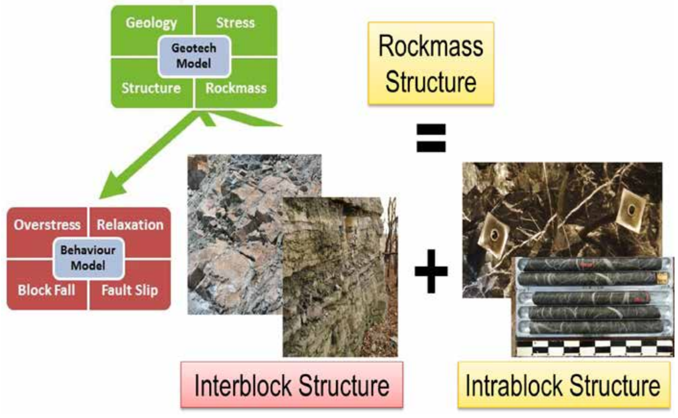
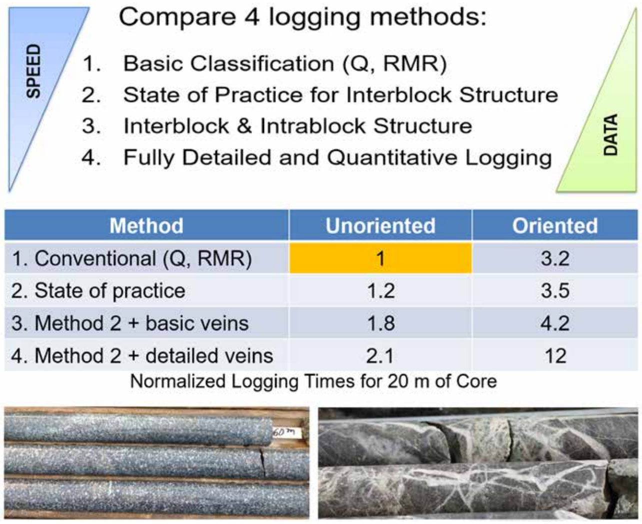


Figure 19: Consideration of Interblock and Intrablock rockmass structure after Day et al



Compare 4 logging methods:

1. Basic Classification (Q, RMR)
2. State of Practice for Interblock Structure
3. Interblock & Intrablock Structure
4. Fully Detailed and Quantitative Logging

Method	Unoriented	Oriented
1. Conventional (Q, RMR)	1	3.2
2. State of practice	1.2	3.5
3. Method 2 + basic veins	1.8	4.2
4. Method 2 + detailed veins	2.1	12

Normalized Logging Times for 20 m of Core

Figure 21: Comparison of logistics(Nickel Rim South Mine and En Cuentro Mine, Chile) of core logging techniques suitable for different analysis levels (after Day et al)

collection (Figure 16), probing techniques (17), and data analysis for validation and calibration as in Figure 18.

Typically, rockmass structure is considered through classic assessment systems such as Q, RMR classification or through GSI characterization and empirical rockmass property assessment, to be used as input into equivalent continuum models. While this sub-project did not result in new numerical tools for the analysis of discrete structure as an alternative to continuum analysis it does explore the engineering promises and pitfalls of emerging tools and provides customized and extended procedures for mining application. Another example of this focus is the work of Day (et al) to characterize complex heterogenous rockmasses containing both joints (physical breaks forming “interblock” structure) and veins (potential breaks forming “intrablock” structure) as illustrated in Figure 19. Such rockmasses are typical of orebodies at all depths although the characterization of mechanical behaviour is critical for deep mining.

Existing and novel analysis tools are capable of simulating the impact of this structure but the existing data collection protocols and modelling practice are in need of upgrading (Figure 20 and Figure 21). Such was the focus and outcome of this work. Figure 22 summarizes two approaches for integrating interblock and intrablock structure into continuum models and Figure 23 compares these two discontinuum models.

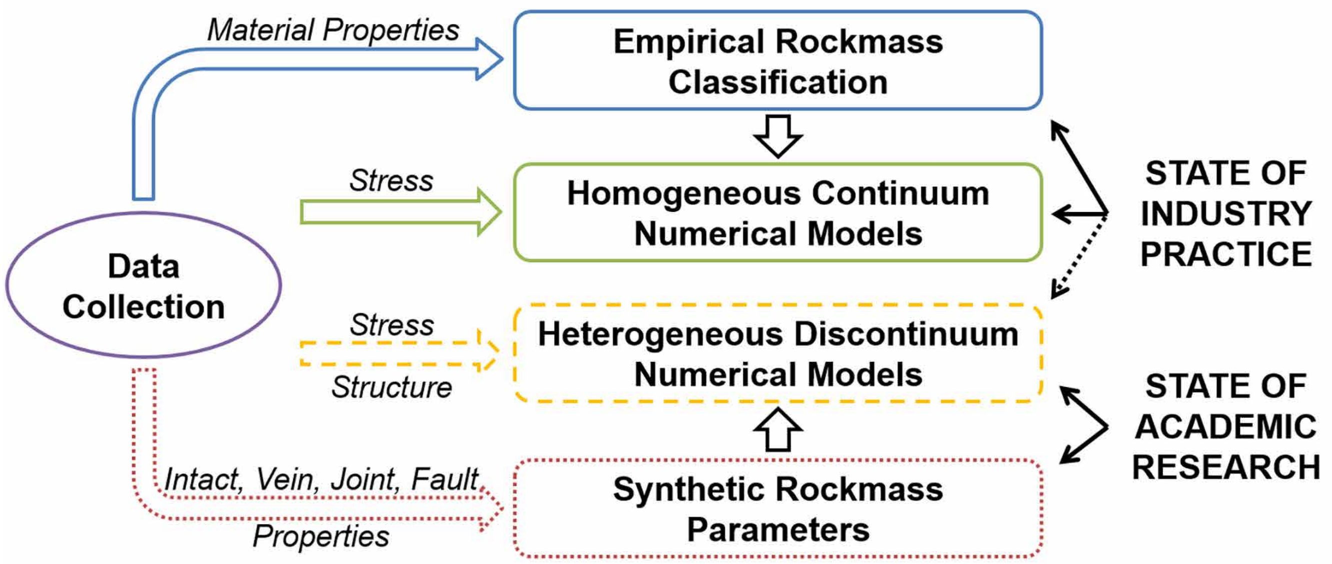
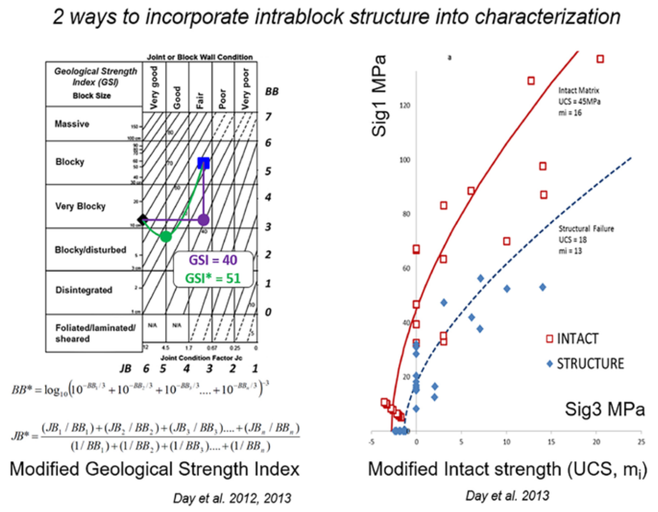


Figure 20: Advancements in data collection linked to analysis complexity (after Day et al)



$$BB^* = \log_{10}\left(10^{-BB_1/3} + 10^{-BB_2/3} + 10^{-BB_3/3} \ldots + 10^{-BB_n/3}\right)^{-3}$$

$$JB^* = \frac{(JB_1/BB_1) + (JB_2/BB_2) + (JB_3/BB_3) \ldots + (JB_n/BB_n)}{(1/BB_1) + (1/BB_2) + (1/BB_3) \ldots + (1/BB_n)}$$

Figure 22: Two approaches for considering intrablock structure in rockmass properties including rockmass adjustment (left) and intact rock adjustment (right) after Day et al

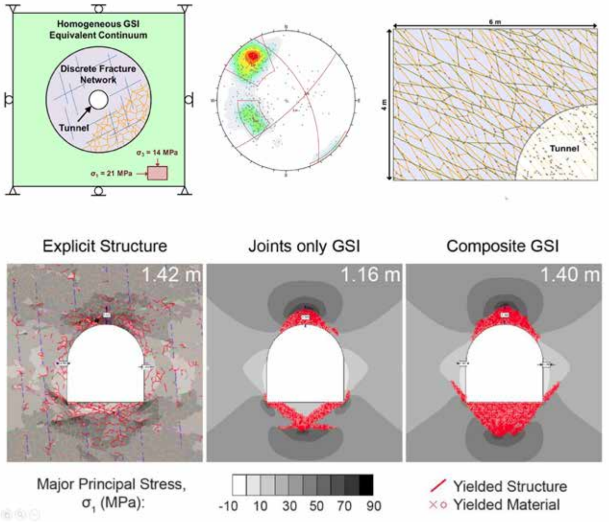


Figure 23: Comparison of continuum approaches with discontinuum modelling (Day et al). White numbers indicate maximum roof yield depth in the three models

One major limitation to conventional geotechnical engineering practice is that it does not always consider the uncertainty in predicting the correct rockmass behavioural mode. The dominant rockmass failure mode is controlled by both the rockmass properties and also by the mining sequence and geometry. Consideration of this uncertainty is a critical step prior to rigorous stability assessment or geo-risk analysis (qualitative or quantitative). Typical failure modes from rockmasses with the same equivalent continuum properties (According to Q, RMR or GSI) are shown in Figure 24 from work by vdP Kraan (et al).

Figure 25 illustrates extended analysis along this theme. Here rockmasses with an identical classification (using the Q system for example) are analyzed using DFN and discontinuum models to simulate the differences in failure mode resulting from variances in the nature of the structure and the stress conditions as well as tunnel geometry. Variances in shotcrete liner response and bolt performance are compared and the sensitivity of design (all have the same empirically designed support) is apparent. This work was extended to provide tools for quantitative assessment of failure mode uncertainty. This analysis can then feed into rigorous quantitative analysis such as that by Langford (et al), as in the examples shown in Figure 26, to round out the contributions in this sub-project.

Practical implications

Consideration of uncertainty is a critical step prior to rigorous stability assessment or geo-risk analysis (qualitative or quantitative) in mines. Tools have been provided for quantitative assessment of failure mode uncertainty.

References Cited

Day, JJ. 2016. The influence of healed intrablock rockmass structure on the behaviour of deep excavations in complex rockmasses. PhD. Thesis, Queen's University.

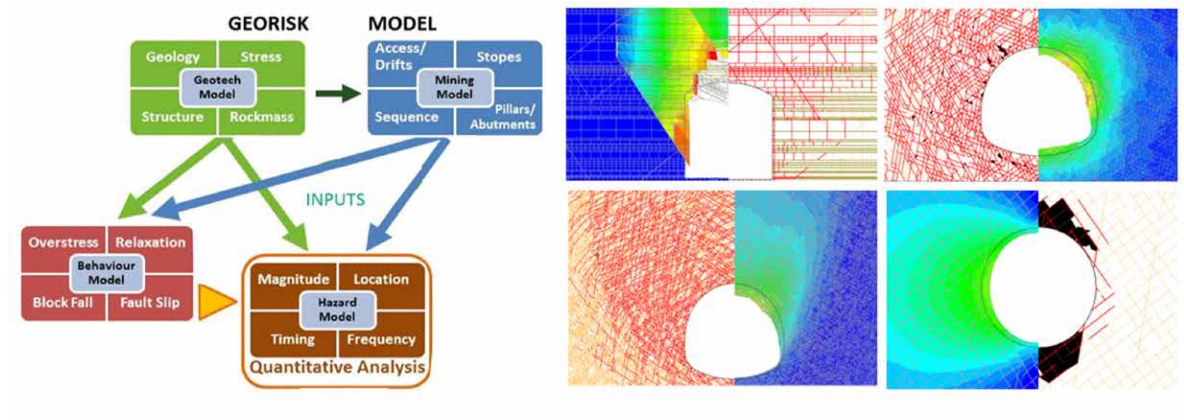


Figure 24: Consideration of rockmass failure mode variability based on structural detail and stress conditions (after vdP Kraan et al). Models at right have the same rockmass classification

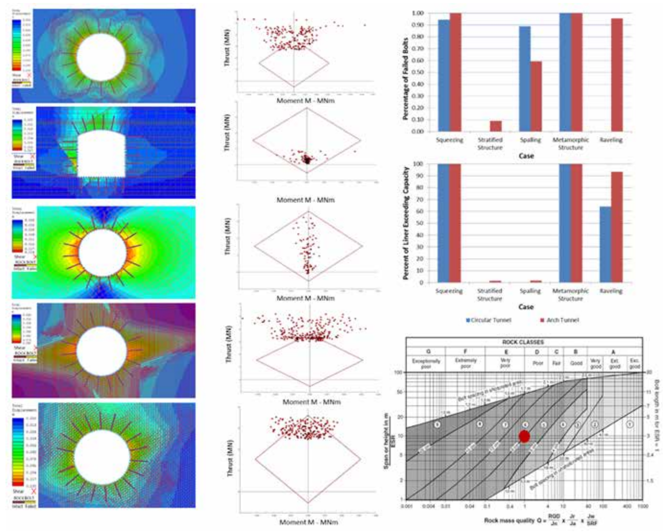


Figure 25: Comparison of support performance for different failure modes (vdP Kraan et al)

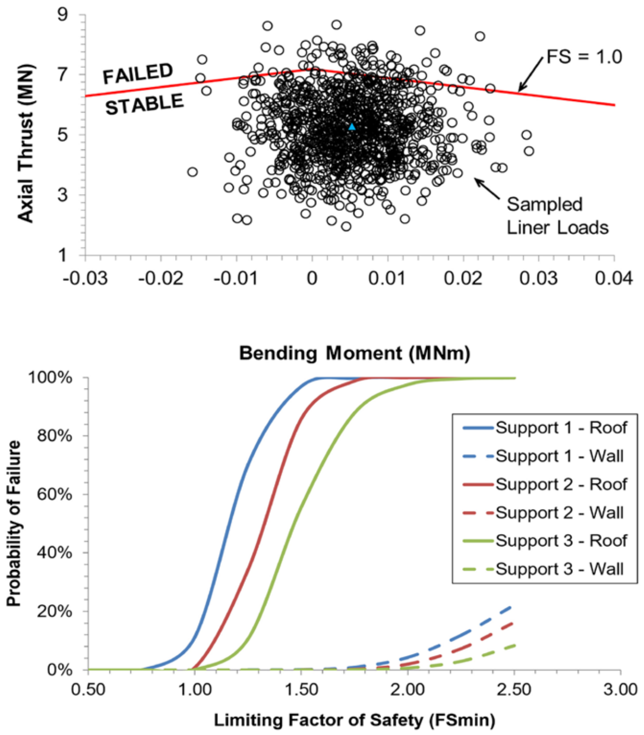


Figure 26: Example of rigorous reliability analysis for shotcrete rock support (Langford et al)

Langford, C. 2013. Application of reliability methods to the design of underground structures. PhD Thesis, Queen's University.

Langford, JC and Diederichs, MS. 2015. Quantifying uncertainty in Hoek-Brown intact strength envelopes, International Journal of Rock Mechanics and Mining Sciences. v74: p.91-102. DOI:10.1016/j.ijrmms.2014.12.008.

Langford, JC and Diederichs, MS. 2015. Reliable support design for excavations in brittle rock using a global Response Surface Method, Rock Mechanics and Rock Engineering. v48:2: p.669-689. DOI:10.1007/s00603-014-0567-z.

O'Connor, S. 2015. Geological models and their influence on geotechnical investigation. MSc Thesis, Queen's University.

Palleske, C., Diederichs, MS., Hutchinson, DJ., and Elmo, D. 2014. Block size distributions as a rockmass classification tool. In Proceedings of the American Rock Mechanics Association conference. June, Minneapolis. Paper 14-7348 8pgs.

Palleske, C.; Lato, M., Hutchinson, DJ.; Elmo, D.; and Diederichs, MS. 2013. Impacts of limited spacing and persistence data on DFN modelling of rockmasses. In: Proceedings: Canadian Geotechnical Conference, Montreal, Quebec.

Van der Pouw Kraan, M. 2014. Rockmass behavioural uncertainty: implications for hard rock geotechnical baseline reports. MASc Thesis, Queen's University.

Selected Publications

Day JJ, Diederichs, MS. and Hutchinson DJ. 2015. Common Core: Core logging procedures for characterization of complex rockmasses as input into geomechanical analysis for tunnel design. Tunnels and Tunnelling. v2015:1: p.26-32.

Day, J., Diederichs, M., Hutchinson, DJ. 2016. Accounting for complex intrablock rockmass structure using the Hoek-Brown criterion and Geological Strength Index Submission April 2016 to Geotechnical and Geological Engineering.

Day, JJ., Diederichs, MS., and Hutchinson, DJ. 2015. Effects of structural contact stiffness on strength and progressive failure of fractures and healed structure. Invited Paper. Geomechanics and Tunnelling. V8:5: p.414-420. DOI: 10.1002/geot.201500027.

Langford JC and Diederichs MS. 2013. Reliability based approach to tunnel lining design using a modified Point Estimate Method, International Journal of Rock Mechanics and Mining Sciences, v60:6: p.263-276. DOI:10.1016/j.ijrmms.2012.12.034.

NOTE: for a complete listing of the authors' publications please go to:
https://scholar.google.ca/citations?user=_79Nqy4AAAAJ&hl=en for Mark Diederichs
www.queensu.ca/geol/hutchinson for Jean Hutchinson

2.2 Rock Mass Strength at Depth

2.2.1 OVERCOMING CHALLENGES OF ROCK MASS CHARACTERIZATION FOR UNDERGROUND CONSTRUCTION IN DEEP MINES

Team:
Laurentian University, MIRARCO – *Drs. Peter K Kaiser* and *Salina Yong* with *Rob Bewick.*

The primary sponsor for this project was Rio Tinto and many collaborators including Florian Amman and Matt Pierce also made to this project.

Abstract

Rock mass characterization for deep underground construction and mining offers many challenges. Kaiser et al. (2015) first describes some of the key challenges when characterizing highly stressed brittle rocks and then offers some guidance on how to arrive at reliable rock mass strength parameters for deep mining applications. Focus is placed on characterization for rock mass strength determination. It is illustrated how the common use of currently available rock mass characterization systems tend to underestimate the strength of rock at depth. Modifications to currently adopted approaches, i.e., the use of Geological Strength Index GSI for rock mass strength determination, are presented.

Context

The context is given by Kaiser et al (2015): For green fields projects, the first challenge of rock mass characterization stems from a lack of access to examine the rock mass at scales larger than typical core sizes (i.e., ~63mm in diameter) until late in the mine design process when the flexibility for making changes is constrained. Hence, rock mass characterization often has to rely on information derived from boreholes alone (e.g., core, logs, etc.). This challenge is lessened when mines are expanded from existing operations. A second challenge is related to changing rock mass behaviour with increasing depth and stress. At depth, the rock is highly stressed, thus closer to failure, and often highly confined which leads to a higher interlocking with elevated strength but more brittleness.

The goals of rock mass characterization are to be tailored to the technical goals of an investigation. For example, for cave engineering, the workflow includes the assessment of five key engineering aspects (Brown, 2007): Caveability; fragmentation; cave performance; extraction level stability; and mine construction (rock support). Each engineering aspect must address specific engineering questions and each design component is influenced by different rock mass behaviour aspects and thus is dominated by different rock mass properties, e.g., the vulnerability to stress fracturing (spalling), the bulking characteristics for flow control and support selection, and the confining stress impact on pillar or support design. An effective rock mass characterization program, including logging, mapping and laboratory testing, thus has to collect and interpret features that are relevant for the above listed purposes. Some of the predominant factors are: block volume of defected and non-defected rock; spalling strength; tensile strength of defects; rock mass modulus; and rock mass strength parameters for application to low and high confinement design aspects.

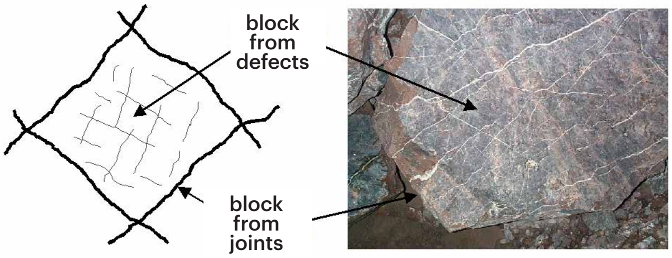


Figure 27: A rock mass with joint bounded blocks that contain veins or other defects; the photograph on the right illustrates a defected rock

Kaiser et al (2015) focus on rock mass characterization of brittle rock to derive rock mass strength parameters for modelling and for use in empirical design. Considering page limitations for this article, the emphasis here is on massive to moderately jointed, heterogeneous rock masses composed of strong intact rock that may contain defects with heterogeneities from strength variation and rock alteration. Such rock masses, as illustrated by Figure 27, consist of heterogeneous rock blocks bounded by block-forming open joints.

Conclusions

Common use of currently available rock mass characterization systems tends to underestimate the strength of highly stressed brittle and often defected rock. It is demonstrated that this is primarily related to flawed interpretation of rock mass characteristics derived from boreholes and laboratory tests without proper consideration of, for example, GSI applicability, sorting failure modes in laboratory testing, and failure modes of rock in underground settings. The key elements for overcoming these inherent flaws are:

- Accounting for the behaviour of massive to moderately fractured brittle rocks at depth and its transition from brittle extensional fracturing in the inner shell to shear dominated fracturing in the outer shell.
- Accounting for the influence of this transition in failure behaviour on the rock mass strength and the limitation in extrapolating low confinement strength to high confinement. Extrapolation can lead to an underestimation of the confined rock strength of up to 50%.
- Conducting laboratory tests to separately characterize both the low and high confining stress range.
- Filtering and grouping laboratory test results according to the

observed failure mode. Failure modes that are affected by defects do not represent the intact rock strength but may give, provided enough test results, a reasonable estimate of the block strength. If failure occurs along a single defect, the strength of the defect can be estimated but this data point must then be eliminated before processing for intact or defected rock strengths using, for example, the Hoek-Brown failure criterion.

- Using face mapping to provide those parameters that cannot be derived from boreholes (e.g., joint persistence, waviness, etc.).
- Characterization derived from both boreholes and underground mapping to help distinguish between defects and block forming joints to determine the applicability of GSI and to eliminate unreasonably low GSI estimates and thus rock mass strength estimates.
- Accounting for the scale of the problem and explicit limitation of the GSI approach. For large block sizes compared to the excavation and/or strongly interlocked joints at depth, failure is dominated by intact rock or rock block failure and the GSI approach underestimates the actual strength in the high confinement range (outer shell).
- Respecting the applicability limits described in this article, i.e., by not applying GSI beyond its range of applicability.

References Cited

Brown, E.T. Block Caving Geomechanics; Issue 3 of JKMRC series in mining and mineral processing, 2002, p.515.

Kaiser P. K., F. Amann and R.P. Bewick 2015. Overcoming Challenges of Rockmass Characterization for Underground Construction in Deep Mines. 13th ISRM International Congress of Rock Mechanics, 10-13 May, Montreal, Canada, p.14.

Selected Publications

Kaiser P. K., F. Amann and R.P. Bewick 2015. Overcoming Challenges of Rockmass Characterization for Underground Construction in Deep Mines. 13th ISRM International Congress of Rock Mechanics, 10-13 May, Montreal, Canada, p.14.

2.2.2 STRENGTH DEGRADATION APPROACH (SDA) FOR ESTIMATION OF CONFINED STRENGTH OF MICRO-DEFECTED ROCKS

Team:
Laurentian University, MIRARCO – *Dr. Peter K. Kaiser* and *Navid Bahrani*

Abstract

It is known that heterogeneity at the grain scale, such as grain geometry and mineral stiffness, as well as grain and grain boundary micro-cracks (called micro-defects) reduces the unconfined compressive strength of rocks. However, how the confined strength of a rock specimen is influenced by the presence of such features is not well understood. In this study, PFC2D and its embedded grain-based model (GBM) is used to investigate the influence of micro-defects on rock strength under unconfined and confined conditions. Previously calibrated GBMs with respect to laboratory properties of intact and heat-treated Wombeyan marble are used to simulate rock specimens with varying micro-defect densities. Micro-defects are simulated in the form of grain boundary frictional cracks. The results of numerical simulations are then used to develop semi-empirical equations that relate the confined peak strength difference of intact and micro-defected specimens. The applicability of the proposed approach, called the Strength Degradation Approach (SDA) for estimating the confined peak strength of micro-defected rocks is demonstrated.

Summary of Findings

The grain-based specimens previously calibrated to the laboratory properties of Wombeyan marble were used to simulate micro-defected rock specimens. Micro-defects were simulated as frictional grain boundaries. The simulation test results including the unconfined and confined strengths of intact and micro-defected grain-based specimens were used to develop a set of semi-empirical equations, which relate the strength reduction from intact to micro-defected rocks to the confinement. Therefore, having knowledge of the unconfined and confined strengths of the intact rock and the unconfined strength of the micro-defected rock, the confined strength of the micro-defected rock can be estimated. The suitability of the Strength Degradation Approach (SDA), was assessed on the results of published laboratory test data from micro-defected rock specimens. It is found that the confined strength of micro-defected specimens can be estimated using the proposed strength degradation approach. Further investigation on the applicability of the SDA for estimating the confined strength of defected rocks with varying defect geometries (i.e., persistence and orientation) and properties (i.e., roughness) is suggested.

Selected Publications

Bahrani N., & Kaiser P.K. 2017. Estimation of confined peak strength of crack-damaged rocks. Rock Mechanics and Rock Engineering, DOI: 10.1007/s00603-016-1110-1.

Bahrani N., Kaiser P.K. & Valley B. 2014. Distinct element method simulation of an analogue for a highly interlocked, non-persistently jointed rockmass. International Journal of Rock Mechanics and Mining Science, 71: p.117-130.

Bahrani, N., and P.K. Kaiser, 2013. Strength degradation of non-persistently jointed rockmass. International Journal of Rock Mechanics and Mining Science, 62: p.28-33.

2.2.3 ROCK MASS STRENGTH DETERMINATION FOR DESIGN OF DEEP UNDERGROUND EXCAVATIONS PRESENTED AS 13TH ISRM LECTURE 2016

Team:
Laurentian University, MIRARCO – *Drs. Peter K Kaiser, Florian Amman* with *Rob Bewick*

Abstract

As explained in the context section for Theme 2, it was hypothesized that standard approaches tend to underestimate confined rock mass strength. This motivated the Laurentian research team to search for reliable rock mass strength parameters for stability assessment and support design as this may save billions of dollars in deep mining.

The challenge for this project was to overcome limitations of available rock mass characterization systems and to obtaining reliable strength parameters for quantitative rock engineering utilizing numerical models. The objective was to provide guidance on how to improve to adopted approaches, i.e., proper use of a modified Geological Strength Index (GSI) including logging, testing, mapping and rock mass rating methods.

The findings were presented in the 13th ISRM lecture and builds on a paper presented at the 2015 ISRM Congress in Montreal. It first describes some of the key challenges when characterizing rock and then offers guidance on how to arrive at reliable rock mass strength parameters for the engineering design of deep underground excavations. The limitations of various available rock mass classification systems, developed for excavation stability assessments and support selection, and rock mass characterization systems, developed for the purpose of rock mass strength determination, are also briefly discussed. The main part of the lecture deals with rock mass characterization for strength determination and on practical implications for the design of deep underground excavations. It is illustrated how common practices in the use of available rock mass characterization systems often tend to underestimate the rock mass strength, particularly when well confined at depth. Improvements to currently adopted approaches are presented to help overcome these challenges. The practical relevance of reliable rock mass strength determination is illustrated by case examples from civil tunneling and mining applications.

Context

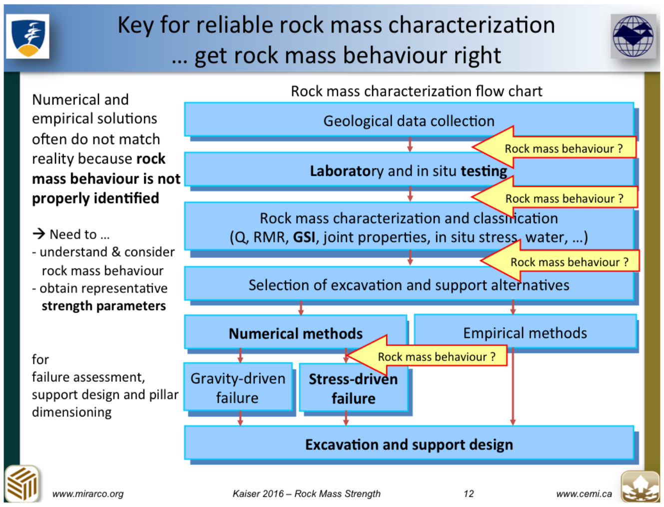
As illustrated by Figure 28, one key element for reliable rock mass characterization is to consider the actual rock mass behaviour such that representative design parameters can be obtained in a quantitative manner.

Site characterization is a gradual process of replacing geomechanics assumptions with facts and data, starting with borehole data in green fields operations, enhancing the data with field mapping and verifying the quantified measures by monitoring.

Methodology

In this lecture, published on-line by ISRM, the author took the position:

- Expertise and experience of world-renowned engineers and scientist such as Brown, Bienawski, Cook, Hoek, Terzaghi, Schoffield, and may more, is truly captured in the GSI-approach and, when applied in a systematic and rigorous manner, serves the rock engineering community extremely well.



Key for reliable rock mass characterization
… get rock mass behaviour right

Numerical and empirical solutions often do not match reality because **rock mass behaviour is not properly identified**

→ Need to …
- understand & consider rock mass behaviour
- obtain representative **strength parameters**

for
failure assessment, support design and pillar dimensioning

Rock mass characterization flow chart

Geological data collection

Rock mass behaviour ?

Laboratory and in situ **testing**

Rock mass behaviour ?

Rock mass characterization and classification
(Q, RMR, **GSI**, joint properties, in situ stress, water, …)

Rock mass behaviour ?

Selection of excavation and support alternatives

Numerical methods

Empirical methods

Rock mass behaviour ?

Gravity-driven failure

Stress-driven failure

Excavation and support design

www.mirarco.org Kaiser 2016 – Rock Mass Strength 12 www.cemi.ca

Figure: 28: Elements of rockmass characterization

- GSI-approach, combined with a curved (Hoek-Brown) failure envelope is well-suited to describe the rock mass behaviour when failing by both extension fracturing and shear rupture.
- When correctly applied within the limits of applicability and verified by field observations, the GSI approach provides the only means to arrive at reliable rock mass strength parameters for use in numerical continuum models and for engineering design.

The GSI approach is "simple" and "scientifically tenable" when combined with rigorous engineering principles and with a sound understanding of rock mass behaviour.

Summary of Findings

The GSI-system does provide a systematic, field tested, semi-empirical means for the determination of rock mass strength parameters when a curved failure envelope properly describes the rock mass behaviour. However, some deficiencies in the GSI system have to be overcome for deep underground construction. Ample evidence is presented in case examples during the lecture to demonstrate that GSI underestimates the strength of rock masses when well confined (> UCS/10 to 15) because the impact of interlock in strong rock is underestimated (joint roughness and over-closure causing dilation and cohesion; non-persistence of even a few % adding much cohesion; and scale and boundary conditions preventing block rotation). It also demonstrated that GSI needs to be established independently by published relations between GSI and RMR or Q.

When GSI is not applicable, as is explained in the lecture, the block strength matters. Near excavation fractured rock blocks can rotate but shear rupture dominates in confined rock. Hence it is necessary to differentiate between inner and outer shell behaviour and to establish a tri-linear failure envelope to describe rock mass behaviour and obtain different engineering properties for the inner and the outer shell.

Practical implications

- Spalling behaviour combined with interlock of strong rock leads to a tri-linear failure envelope for rock involving stress-fracturing.
- Inner and outer shell characteristics differ drastically
- Inner shell is dominated by “Spalling strength” of rock mass
- Outer shell is dominated by “confined rock mass strength”.
- Strength of blocks bound by open joints controls rock mass strength when GSI is not applicable.
- Strength of defected blocks controls rock mass strength when GSI is applicable.
- The following steps must be followed to obtain reliable rock mass strengths:
- Independently obtain GSI considering all relevant factors.
- Obtain intact strength sci from triaxial tests after filtering for defected and structurally dominated sample failures.
- If defected rock, obtain block strength sbl and replace sci by sbl in GSI equations
- Check for GSI applicability (scale, trace lengths, interlock, etc.).
- If GSI is not applicable, obtain separate rock mass strength parameters for low and high confinement range (Tri-linear envelop for inner/outer shell transition).
- Fit tri-linear rock mass strength envelope for constitutive model in numerical modelling code
- Establish post-peak strength (not covered in detail) considering strain-dependence of ultimate or residual test data.

Conclusions

Current rock mass characterization systems tend to underestimate confined strength of highly stressed rock and the modified GSI approach presented in the lecture provides a means for reliable rock mass strength determination for green fields situations as long as GSI applicability limits are not violated (GSI is typically not applicable for GSI > 60-70 obtained from logging).

If GSI is not applicable, an alternate approach, presented in this lecture, is followed but the outcomes of classification always need to be verify by field observations. The key to overcome flaws in rock mass strength determination are:

- Account for “inner and outer shell” rock mass behaviour
- Do not extrapolate from inner to outer shell behaviour
- Conducting laboratory tests for both low and high confining stress range
- Filter laboratory data according to the observed failure mode
- Only use intact breaks for strength determination by GSI
- Integrate into observational approach:
- Start with experience-based assumptions but replace them with data and verified facts.

Figure 29 illustrates this flow:

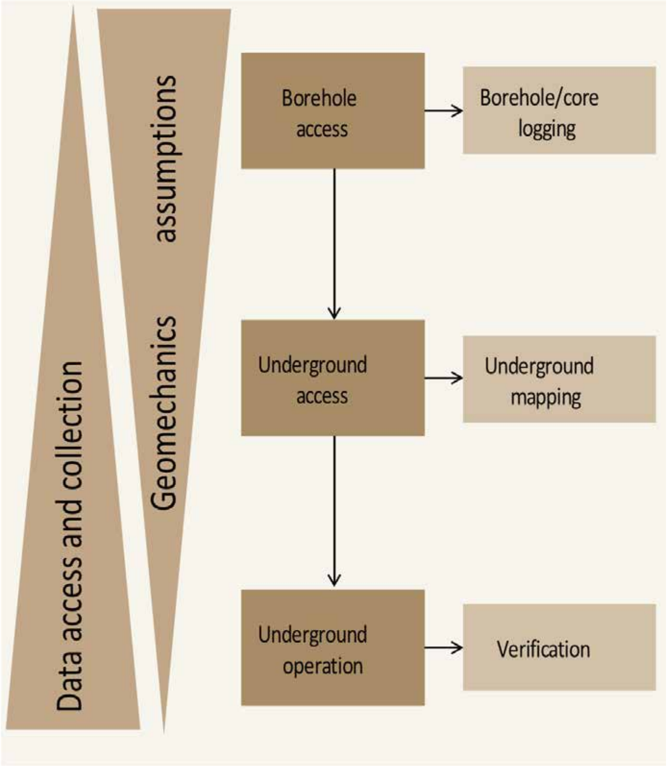


Figure: 29. Overcoming flaws in rockmass characterization

Selected Publications

Kaiser PK, 2016. Challenges in Rock Mass Strength Determination for Design of Deep Underground Excavations. Published as ISRM on-line lecture. 45 min., 70p.

Kaiser P. K., F. Amann and R.P. Bewick 2015. Overcoming Challenges of Rock mass Characterization for Underground Construction in Deep Mines. 13th ISRM International Congress of Rock Mechanics, 10-13 May, Montreal, Canada, 14p.

Bewick, R. P., F. Amann, P. K. Kaiser and C. D. Martin 2015. Interpretation of UCS test results for engineering design. 13th ISRM International Congress of Rock Mechanics, 10-13 May, Montreal, Canada, 14p.

NOTE: For a complete listing of the authors’ publications please go to:
https://scholar.google.ca/citations?user=BM3Ds4EAAAAJ&hl=en for Peter K Kaiser
https://scholar.google.ca/citations?user=cIscgCkAAAAJ&hl=en for Amann and
https://scholar.google.ca/citations?user=868X_2EAAAAJ&hl=en for Bewick

2.3 Automated Characterization and Change Detection within 3D Digitally Scanned Virtual Excavation Boundary and Borehole Models to Quantify the ROCKMASS Deformation Response to Mining Over Time

Team:

Queen's University – *Drs. Mark S. Diederichs* and *Jean Hutchinson* with students: *Gabe Walton, Jenn Day, Connor Langford and, Ioannis Vazaios, Cortney Palleske, Dani Delaloye, Felipe Duran, Michelle vdP Kraan* and *Shaun O'Connor.*

Context

Modern scanning techniques, including photogrammetry and LiDAR (laser scanning), offer the opportunity to improve data input for rockmass characterization, mine geometry verification and change detection and displacement monitoring over time (Figure 30). Early work based on past Queen's developments drove further optimized techniques (Lato et al and Vazaios et al) for structural data extraction from 3D point-cloud data (Figure 31).

Methodology

Improved scanning protocols as well as purpose-generated statistical techniques to accurately and precisely measure and map displacements and physical changes (fallout) during tunnel/mine life were investigated and developed (Figure 32).

Summary of Findings

The results of this research and development were trialed, validated and optimized through field studies initially at Coleman Mine (Figure 33) through the work of Delaloye (et al) and later at Creighton Mine (Figure 34) through the work of Walton (et al). Many technical and logistical challenges (timing, mesh interference, atmospheric conditions) were addressed in this work. In the case of Creighton, LiDAR change mapping correlated well with detailed deformation monitoring using extensometers (section 4.2). More than 12 graduate students benefited from their direct or indirect involvement in this work (9 are highlighted here).

Practical Implications

This work has built on existing expertise in the use of laser scanning (LiDAR) and data processing to extract geotechnical data, progressing to the use of LiDAR data to accurately determine and map deformations and change within underground excavations. This has led to advances in our ability to create more realistic models for analysis, and in our capacity to assess the inputs and interpret the outputs of such models for design and Geo-risk reduction. This represents a significant step forward in the use of field data for model validation and calibration, leading to increased reliability of design modelling.

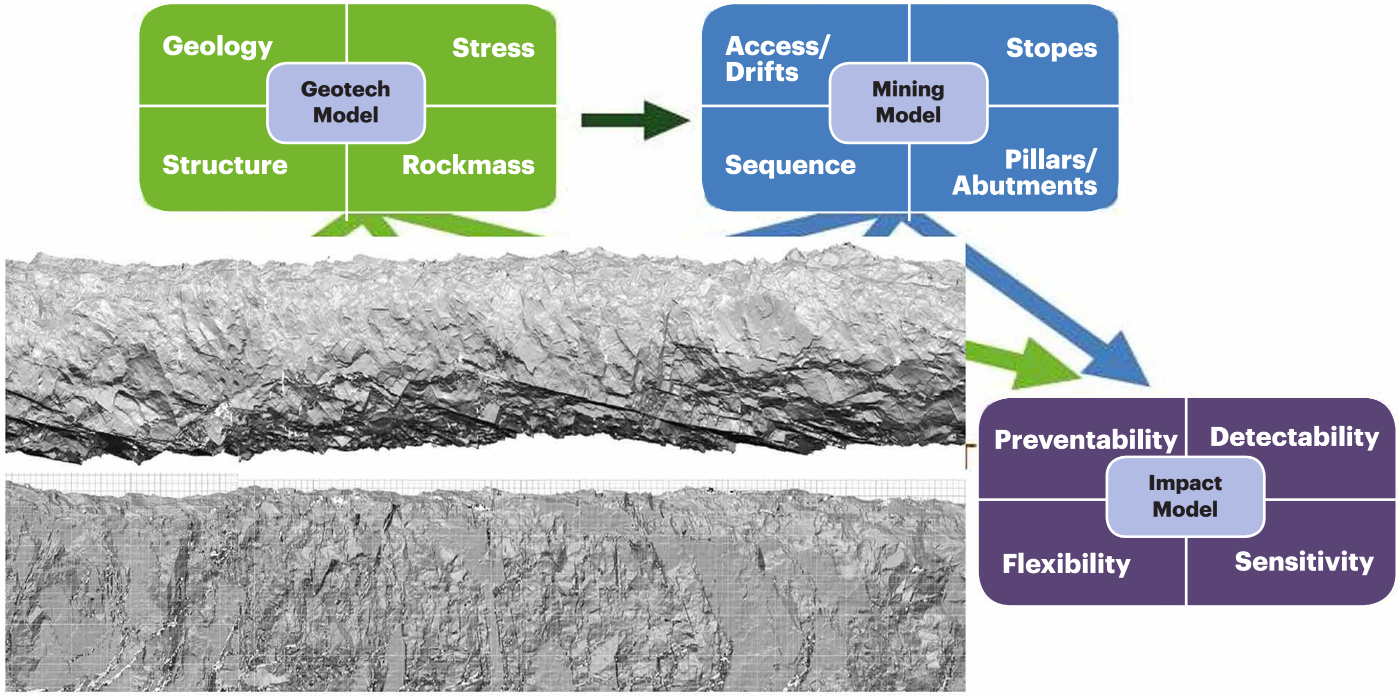


Figure 30: Rockmass change detection using LiDAR (3D mesh of a tunnel – Vazaios et al).

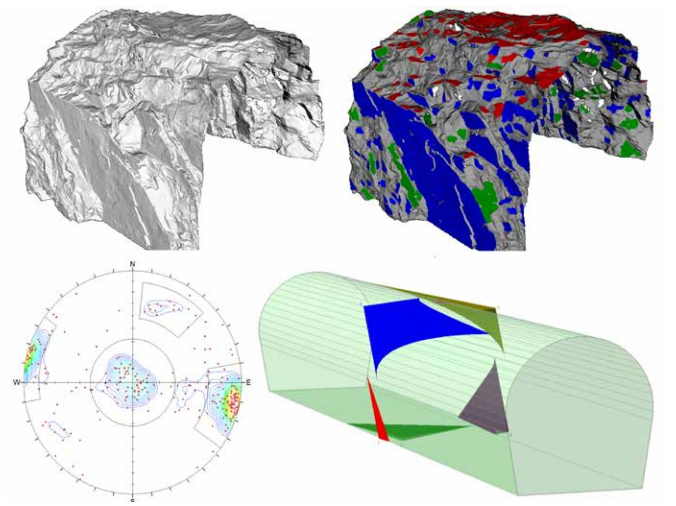


Figure 31: Automated structural mapping from LiDAR (Lato et al and Vazaios et al)

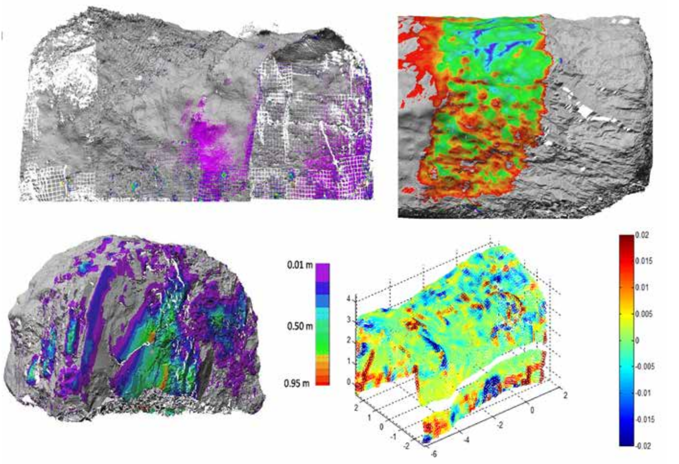


Figure 32: Change and displacement detection over time (Delaloye et al and Walton et al)

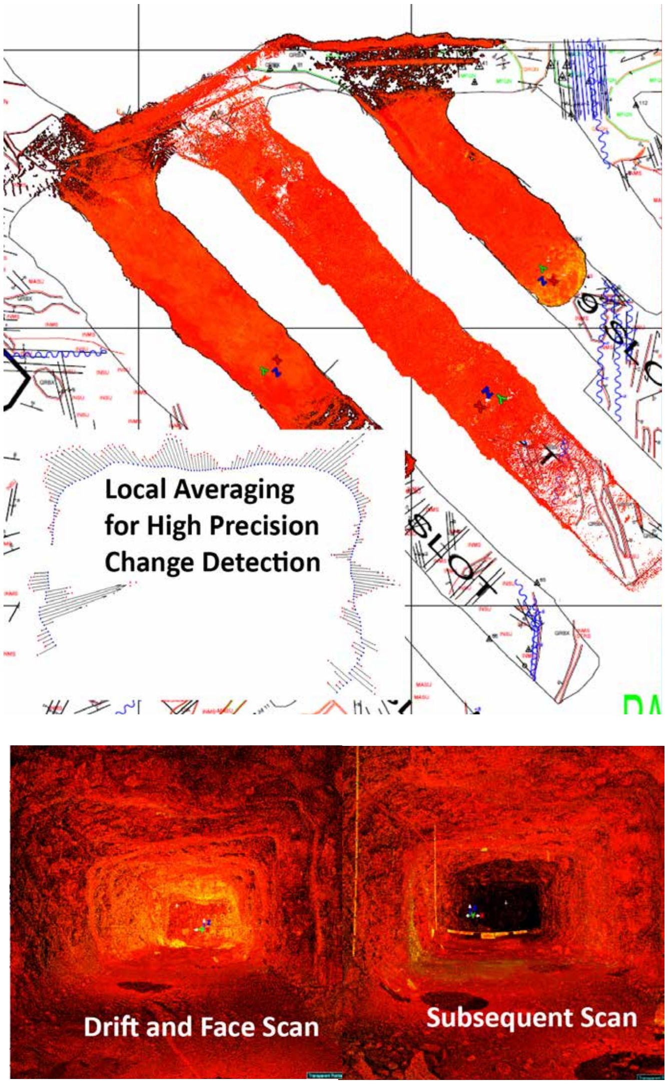


Figure 33: LiDAR scanning and change detection at Coleman Mine (Delaloye, Walton et al)

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Delaloye, D., Hutchinson, DJ. and Diederichs, MS. 2012. A New Workflow for LiDAR Scanning for Change Detection in Tunnels and Caverns. In Proceedings of the 46th American Rock Mechanics Association Geomechanics Symposium. Chicago June 2012. ARMA 12-354. 10p.

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Walton, G., Delaloye, D., and Diederichs, MS. 2014. Development of an Elliptical Fitting Algorithm for Tunnel Deformation Monitoring with Static Terrestrial LiDAR Scanning. Tunnelling and Underground Space Technology. v43: p.336-349. DOI 10.1016/j.tust.2014.05.014.

Walton, G., Diederichs, MS., Weinhardt, K. Delaloye, D. Lato, M. 2016. Change detection in drill and blast tunnels from point cloud data. Submitted. October 2015 to Rock Mechanics and Rock Engineering.

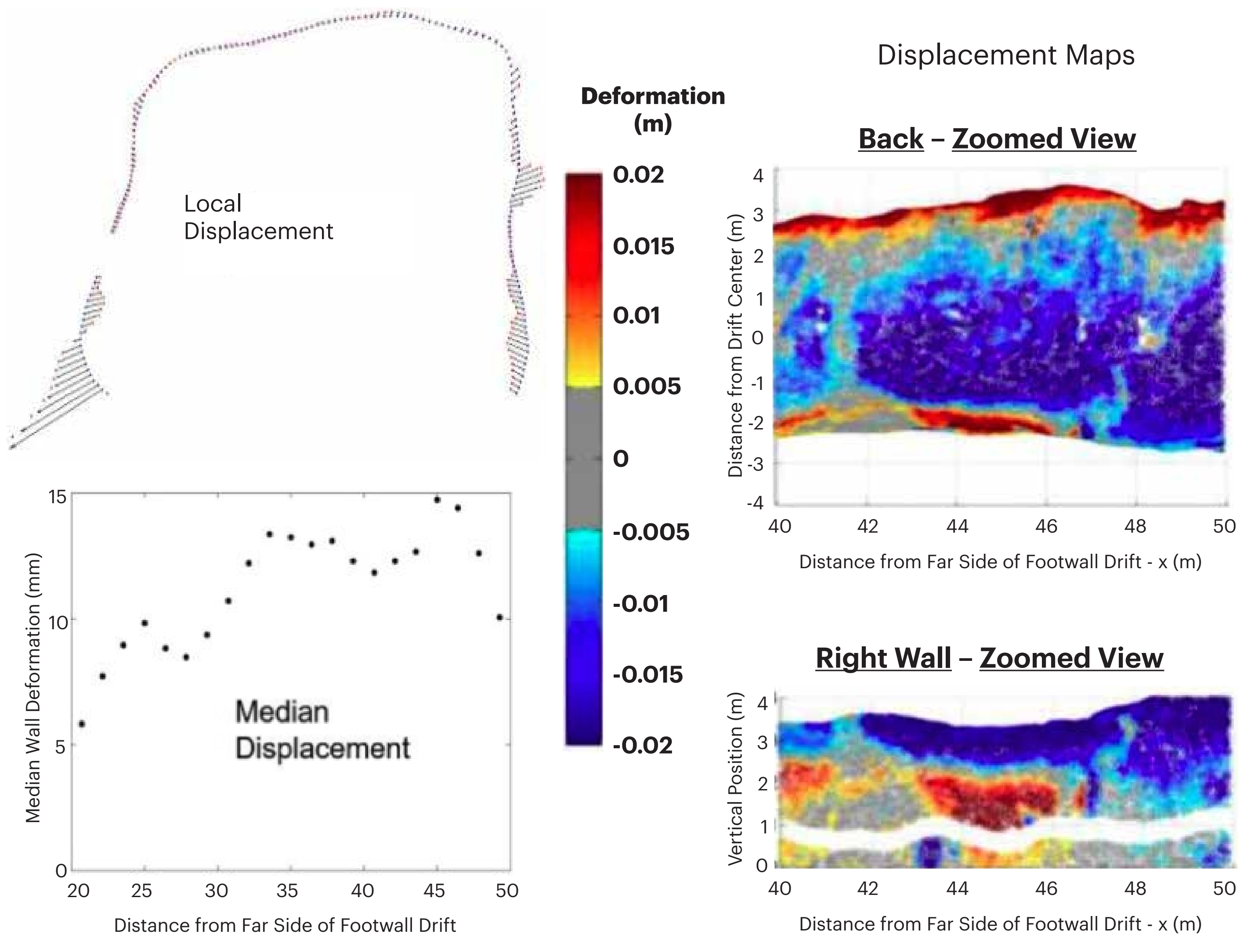

Figure 34: LiDAR scanning and displacement monitoring (Walton et al)

Selected Publications

Delaloye, D., Diederichs, MS., Walton, G., and Hutchinson, J. 2014. Sensitivity Testing of a Newly Developed Elliptical Fitting Method for the Measurement of Convergence in Tunnels and Shafts. Rock Mechanics and Rock Engineering. v48:2: p.651-667. DOI 10.1007/s00603-014-0566-0.

NOTE: for a complete listing of the authors' publications please go to:
https://scholar.google.ca/citations?user=_79Nqy4AAAAJ&hl=en
for Mark Diederichs
www.queensu.ca/geol/hutchinson for Jean Hutchinson

Acknowledgements

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SUMIT #3

Seismic Processing and Stress Modeling

3.1

Borehole Logging and Petrophysical Lab Studies

3.2

Towards Far-Field Stress Calibration Through Measuring Tunnel Deformation with LiDAR Imaging

3.3

Real-Time Monitoring of Geophysical Data in Deep Mines

3.4

Remote Geophysical Monitoring of Stress/Strain

3.5

Dynamic Support Research for More Effective Mine Design

Seismic Processing and Stress Modeling

Objectives

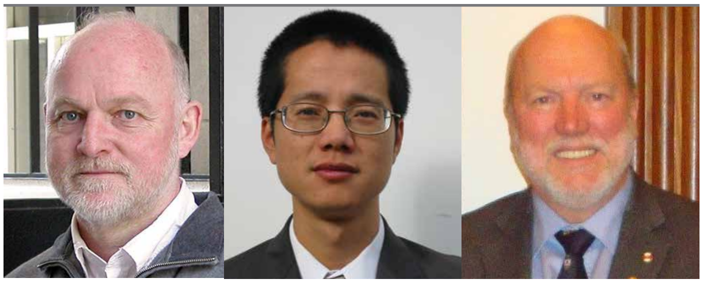
Modern geophysical techniques permit “looking into the ground” and processing of “fantastic” data sets. This theme was therefore intended to advance the state-of-the-art in both technology development and utilization for purposes assisting in deep mining, not just exploration. Specifically, the research conducted under this theme focussed on:

- Developing a method to estimate the orientation and magnitude of excavation-induced stress changes though back-analysis of tunnel deformation measurements for use in far-field stress calibration.
- Development of 3D geophysics to more effectively “see” between boreholes by testing borehole/cross hole radar, seismics, EM and DC/IP for detection/anticipation of the capabilities of ground conditions and adverse geological structures including cross-correlation of borehole geophysics with full waveform sonic logs (k, μ, Q) and borehole televiewer logs.
- Extension of 3D Seismic tomography with improved velocity models.
- Study of seismic wave propagation mechanisms and identification of key factors that alter ground motion in complex rocks in deep underground mines; in particular, PGV (peak ground velocity) distribution along drifts and near the excavation boundaries. The latter is focused on providing a methodology for improved rock support design.

3.1 Borehole Logging and Petrophysical Lab Studies

Team:

University of Toronto – *Drs. Bernd Milkereit; Kaiwen Xia, Paul Young,* and students: *Kun Guo, Ramin Saleh, Mario Tibbo, Ken Nurse, Anisa Kassam, Jinfei Cai, Alex Carey, Rakesh Srirajaraghavaraju* and *Patrick Kanopolous.*



Title: Dr. Bernd Milkereit, Dr. Kaiwen Xia, Dr. Paul Young
Collaborators on team: Dr. Douglas Schmitt, University of Alberta, Dr. Wei Qian, Geoserve, Vale, Sudbury Integrated Nickel Operations – A Glencore Company

Abstract

Work has lead to the development of a new methodology, instrumentation, and work flows for long-term, detailed study of the stress, strain, and time-lapse geophysical responses of a well-characterized volume of rock in a deep mine. For the SUMIT project, the selected test site was at a seismically active mine at 1-2 km depth with dedicated boreholes (I) to characterize the 3D rock volume through core, logging, and geophysical imaging in order to quantify the initial stress state and physical properties and then (II) to monitor the temporal and spatial variations of these extrinsic conditions and stress and associated physical properties within the rock volume over the following three years. In the process, we developed the first dynamic (time variant, time stamped) 3D deep mine model through the integration of geology, physical rock properties, infrastructure, production and backfill. Lab measurements on drill core samples confirmed that most physical properties of crystalline rocks are highly stress dependent.

The SUMIT data showed that P- and S-wave velocities and electrical properties are linked directly to changes in stress due to the reduction of fracture porosity. For rock mass characterization, borehole geophysical data provide reliable estimates of geotechnical parameters such as dynamic Young’s modulus at elevated in situ stress levels. Borehole televiewer data map the direction of minimum and maximum horizontal compression. The continuous borehole geophysical and televiewer data confirm the important role of geology, as elastic moduli and concentration of azimuthal stress vary with lithology. At the SUMIT test site, the first time-lapse geophysical surveys for stress monitoring were conducted in two slim holes in the vicinity of the active mine at one to two km.

Context

Knowledge of the magnitudes and directions of stress in the earth is fundamental to our understanding of plate tectonics, the formation of faults, and the initiation and propagation of earth quakes at all scales (Duff et al., 2011). Stress controls local failures within underground structures and stress can influence the geophysical and hydrological transport properties in the subsurface. For example, the anisotropy of the principal stresses within the earth affects the anisotropy of electrical resistivity, seismic wave velocities, and permeability.

Tectonic stress and stress redistribution from the process of mining can lead to unintended consequences as induced stresses interact with existing fractures and faults (Duff et al. 2010). Thus, monitoring and understanding stress variations in space and time are of great importance. As physical rock properties, such as compressional wave velocities and conductivities are sensitive to stress changes, continuous monitoring of 3D controlled source geophysical data may provide new insights into spatial and temporal variations of stress at the SUMIT test site.

Conversely, remotely measuring these properties using geophysical methods might provide methods to better understand stress distributions. Ideally, one would hope to be able to map the stress tensor in 3D within a volume of the earth. This is not so easily achieved; and often only indicators of stress are all that are available. Such indicators include geology (the attitudes and directions of igneous dikes and sills) and the tension and compression axes of microseismic focal mechanisms. For better site characterization, a detailed 3D mine model was built to investigate the role of geology and temporal changes due to infrastructure, production and backfill on seismic wave propagation and how temporal variations within the 3D mine model affect and distort microseismic data.

The University of Toronto's SUMIT research project's primary technical goal is to characterize the rock mass through petrophysical and borehole geophysical studies and then carefully monitor the long-term variations of geophysical properties with time in a large volume of rock at depths of 1.3 km to 2 km. To achieve this goal, time-lapse borehole geophysical studies were conducted and multi-sensor arrays were deployed to assess the magnitude of spatial and temporal changes of the physical rock properties.[4]

3.1.1 3D/4D SEISMIC WAVE PROPAGATION IN A DEEP MINE

Context

Currently, only microseismicity is used as a proxy for stress distributions in deep mines. Mine development imposes stress changes in the surrounding rock that typically induce or trigger seismicity with a wide range of magnitudes. In an active, deep underground mine site, controlled production blasts and microseismic monitoring are used to better understand the process of rock fracturing, fault slip, and rock deformation. The production blasts and microseismic events typically have high frequency content up to a few kHz that leads to small wavelengths on the order of a few metres in hard rock environments.

Contrasts in the elastic-wave velocities and densities of rocks are the most important physical properties influencing the overall seismic response. The seismic wave velocities depend on the in-situ elastic properties, which in crystalline metamorphic and igneous rocks are controlled by mineralogical content, damage, stress, in-situ fractures, pressure, and saturation. A deep mine site is a very complex medium for the propagation of the seismic waves due to the presence of many strong velocity and density contrasts in the existence of small-scale heterogeneities. This SUMIT study was conducted at Nickel Rim South (NRS) mine, in Sudbury, Ontario, Canada. The NRS mine site offers an interesting geological setting for fundamental research into time-lapse monitoring of seismicity, stress, and stress dependent physical properties at depth.

At NRS, the major heterogeneities with the strongest elastic contrasts include massive ore bodies, tunnels, stopes and infrastructure (Figure 35). These contrasts are varying in size and elastic properties and will affect seismic wave propagation and thereby will impact microseismic waveform data in terms of amplitude, phase and travel time.

Accurate estimation of amplitudes and travel times of seismic events are critical parameters that have to be determined at various

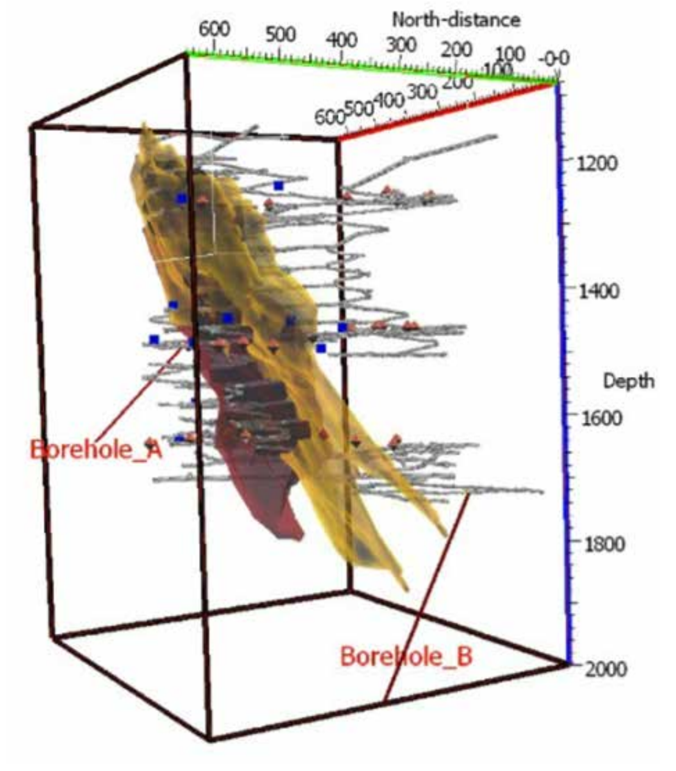


Fig 35: Geological 3D used mine model including ore bodies (yellow and red), tunnels (gray) and stopes (red). Red and blue cubes show 3D distribution of 3C geophones and accelerometers, respectfully at the mine used in this study. Boreholes A and B are used for time-lapse geophysical measurements. Core samples from boreholes A and B are used for petrophysical and geotechnical studies

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[4] *The work which focused on change detection is reported on in the SUMIT #4 section*

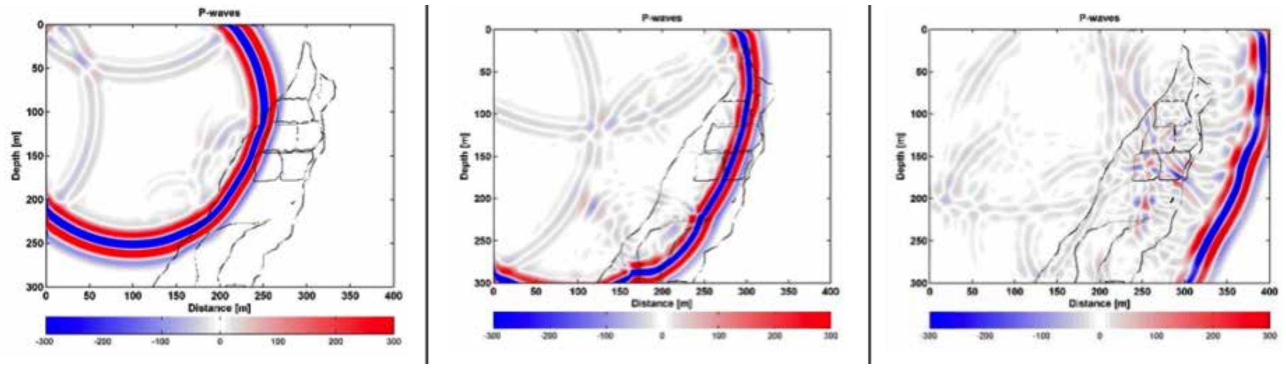

Figure 36.1.: P-wave snapshots here represent a simulation of a blast located on the top left corner of model, no mining activity in designated stope areas. The used source central frequency is 200Hz

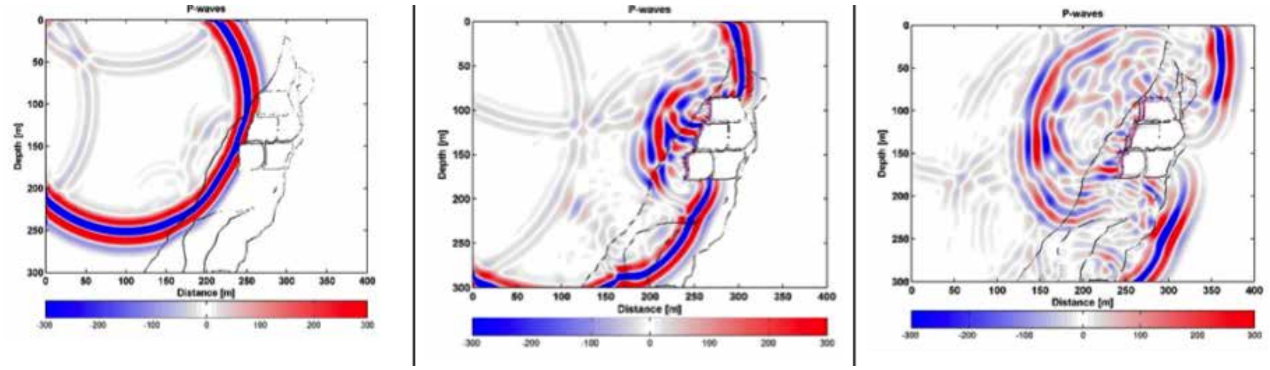

Figure 36.2:. The mine model shows ore bodies mined out and the voids remaining in stope areas. At a later time, note strong reflections and severe attenuation of seismic signal

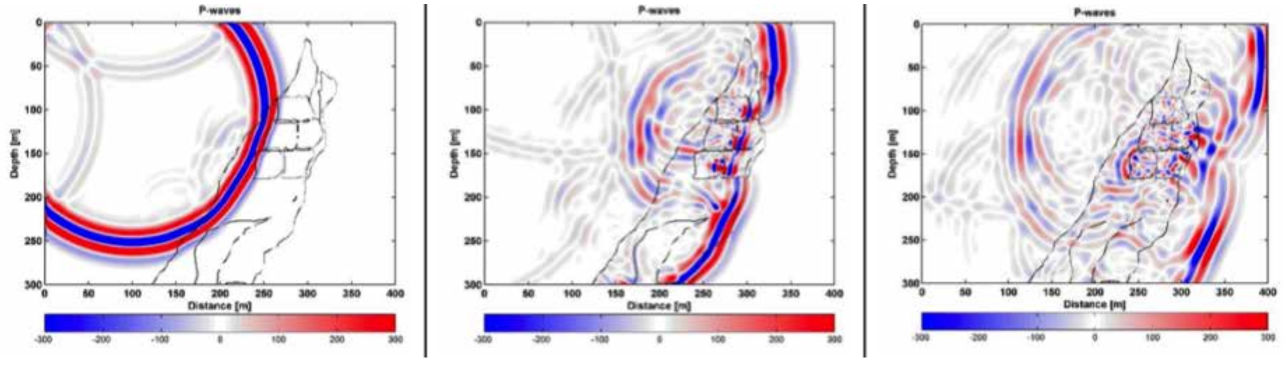

Figure 36.3:. Snapshots represent the mine model afer orebodies are mined out and voids cemented backfilled. Backfilled areas generate strong reflection, significant travel time changes, and lateral amplitude variations

locations in a mine. These parameters are useful in preparing risk maps and to understand better the process of spatial and temporal stress distributions in a mine. As a result, seismologists are trying to use more detailed models and more advanced numerical methods to simulate the propagation of seismic waves more accurately for such complex mediums. Currently simple constant-velocity models used for microseismic monitoring at active mines cannot explain the observed complexities in scattered seismic waves. The travel time and amplitude of seismic waves derived from the conventional constant velocity models are inadequate for such a heterogeneous medium.

Since the complexity of seismic wave propagation can affect the distribution of energy significantly, the use of a more accurate model is required to predict ground motion. For example, the conventional empirical method used to calculate peak particle velocities and accelerations (PPVs/PPAs) tends to underestimate the intensity of seismic waves in stopes or areas close to blast sites, which could be predicted if a more realistic model was implemented. To improve upon this, deep mine modeling of elastic seismic wave propagation would require detailed 3D petrophysical, infrastructure, production and backfill data.

With the availability of detailed 3D geological and physical rock property data at the NRS mine and advances in computational seismology, in combination with access to high performance computing facilities, a solution for the propagation of seismic waves at the mine size scale is achievable.

Methodology

For the SUMIT project at NRS, in order to model the effects of small-scale, high elastic contrasts on the propagation of seismic waves within a time-varying 3D mine model, a full elastic/viscoelastic finite difference software package (Bohlen, 2002) is used.

To visualize the effects, a vertical 2D section from a 3D detailed petrophysics model of an underground mine was used here and illustrated in Figure 35. The model dimension is 640 m x 640 m x 480 m in x, y and z direction respectively. A Ricker wavelet signal with a varied central frequency ranging from 50 Hz to 1000 Hz was used. The model is discretized into structured grids with the grid cell sizes varied between 0.25 m to 2 m, chosen based on the selected source central frequency for each simulation.

Summary of Findings

Modeling results in Figure 36 clearly show the complexity of the propagating waves in mine models. They show that using a homogeneous velocity background model is inaccurate in deep mines. The mine is a very heterogeneous medium resulting in a complexity of wave propagation with variations in amplitude, travel time, and phase. The significance of these effects strongly depends on the size, shape, petrophysical properties and the frequency content of seismic sources. For those simulations with a central source frequency below 100 Hz, the scattered waves from the ore body or stopes do not exhibit significant changes in polarity of first arrivals (not shown in Figure 36), although their travel times are affected.

However, at frequencies above 100 Hz, energy is more scattered and amplitude variation is more significant. Moreover, at some locations the presence of shadow zones behind the contrast is observed. The modelling results verify the amplification effects on both P and S wave amplitudes at regions with a high Vp/Vs ratio (such as voids or cemented backfill). Sources with higher frequency content displayed changes in polarity and/or creation of shadow zones.

The cemented backfill region, a low velocity region, has trapped the energy, with this effect being more dominant for S-waves. This region is mainly responsible for changes in amplitudes and travel times, and clearly shows the need to update the 3D petrophysical model of the active mine site.

Based on the location of the 3D microseismic array sensors (Figure 35) in respect to production blasts and ore body, some seismic recordings show significant travel time deviations (of 10-15 ms) from those using a half space velocity model. Such a travel time difference in hard rock environments would translate to spatial location errors of approximately 60-90 m.

Practical Implications

The detailed 3D/4D seismic modeling study conducted at the NRS mine identifies three potential problem areas for microseismic studies. First, seismic wave propagation is affected by the size, shape, petrophysical properties and the frequency content of seismic sources. Second, the modelling results verify the amplification effects at regions within the 3D mine model. And finally, the presence of voids or backfill in 3D mine models will result in changes of polarity and/or creation of shadow zones. The existence of such complexity in recorded seismic waveforms suggest the necessity for further consideration of such effects in the determination of focal mechanisms and full moment tensor inversions and the placement of three-component seismic component sensors within a microseismic acquisition system.

3.1.2 PHYSICAL ROCK PROPERTY DATA AND FRACTURE POROSITY AFFECTED BY STRESS

Context

The World Stress Map data is determined by stress indicators including earthquake focal mechanisms, in situ measurement in mining, oil and gas boreholes as well as the borehole cores, and geologic data. Unfortunately, these measurements are not only infrequent but sometimes infeasible, and do not provide nearly enough data points with high accuracy to correctly infer stress fields in deep mines. Improvements in stress measurement of the Earth's crust are fundamental to several industries such as oil and gas, mining, nuclear waste management, and enhanced geothermal systems.

Quantifying the state of stress and the geophysical properties of different rock types is a major challenge in geophysical monitoring of deep mines. Most stress measurement techniques involve either the boreholes or their cores, however these measurements usually only give stress along one axis, not the complete stress tensor. The goal of this project is to investigate a new method of acquiring a complete stress tensor of the in-situ stress in the Earth's crust.

Methodology

A comprehensive and systematic petrophysical baseline study was conducted on drill core samples from two deep boreholes (marked A & B in Figure 35) located in the NRS mine. These boreholes are approximately 400 m long with NQ diameters and extend between depths of about 1300 – 1600 m and 1700 – 2000 m. Borehole geophysical logging surveys were performed in both boreholes, between October 2013 and July 2015, in order to perform a time-lapse analysis of the geophysical changes in the mine. These multi-parameter surveys include caliper, full waveform sonic, televiewer, chargeability (IP), and resistivity.

Laboratory experiments have been performed on borehole core samples of varying geologies from each borehole. These experiments (conducted under room conditions, under uniaxial stress conditions, and under controlled confining stress conditions) measured the geophysical properties including elastic modulus, bulk modulus, P- and S-wave velocities, and density (Figure 37). In addition, Brazilian disc (BD) sample tests were conducted. Figure 38 shows a BD test with tensile failure along the line of loading.

Using cores from each borehole, P, S1 and S2 velocity measurements were made using the Floor Standing Acoustic Measurement System (FSAS) (ErgoTech Ltd). Seismic velocity measurements were performed with hydrostatic stresses by loading and unloading the samples from 1-60 MPa, remaining within the elastic/viscoelastic regime of the rock samples.

Summary of Findings

Constructed seismic velocity-stress curves show that the unloading velocities are higher than the loading velocities, as the cores have not had time to relax and the cracks are still closed. Figure 37 shows the compressional and shear wave velocity curves as a function of hydrostatic pressure for a sample taken from borehole B. Derived products include dynamic Young's modulus and fracture porosity as a function of hydrostatics pressure (Kaksor et al., 1999; Asef and Najibi, 2013). The parameters of empirical expression between seismic wave velocity and pressure are fitted based on the laboratory test data. The results show that compressional wave velocity, shear wave velocity, bulk modulus, and Birch's seismic parameter exponentially increase with axial pressure. In contrast, rock porosity and crack density parameter exponentially decrease with axial pressure. With the increase of axial pressure, the value of rock porosity gradually decreases to less than 1% at pressures greater than 35MPa. The microcrack porosity within drill core samples from NRS is highly sensitive to stress variations. Note that the very low effective porosities at elevated pressures are important for the interpretation of the time-lapse geophysical monitoring data in the NRS mine.

Ongoing studies of drill core samples from NRS include controlled triaxial stress tests. The pieces of apparatus used for this project are geophysical imaging cells capable of hydrostatic stress (σ1 = σ2 = σ3), differential stress (σ1 > σ2 = σ3) and the unique true triaxial stress (σ1 > σ2 > σ3). Velocity surveys can be acquired along all three axes, and therefore the effects of σ1, σ2, σ3 on the velocity-stress curve can be obtained. These geophysical cells are being used to reproduce the borehole P- and S-wave velocities by altering the differential stress, allowing for the unique position of determining the stress tensor.

Practical Implications

Physical rock property studies on selected drill core samples confirm stress dependent geophysical parameters and very low fracture porosities (of < 1% at stress > 35 MPa). The data showed that P- and S-wave velocities are linked directly to changes in stress due to the reduction of fracture porosity.

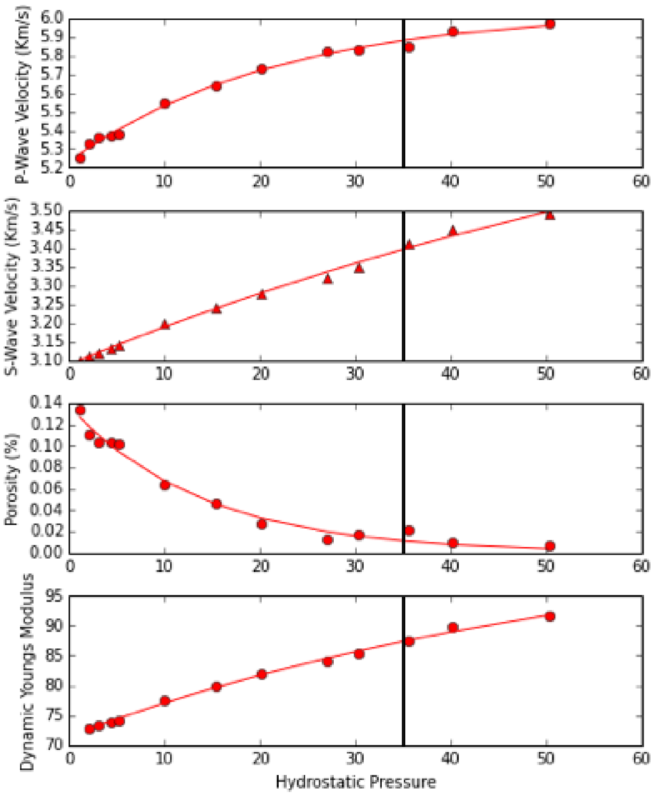

Figure 37: Lab measurements on core samples provide compressional and shear wave velocity as a fraction of universal stress. Fracture porosity decreases with increasing stress. Dynamic Young's Modulus derived from P- and S-waves increases with increasing hydrostatic pressure. Note the low effective porosities for hydrostatic pressures greater than 35 MPa

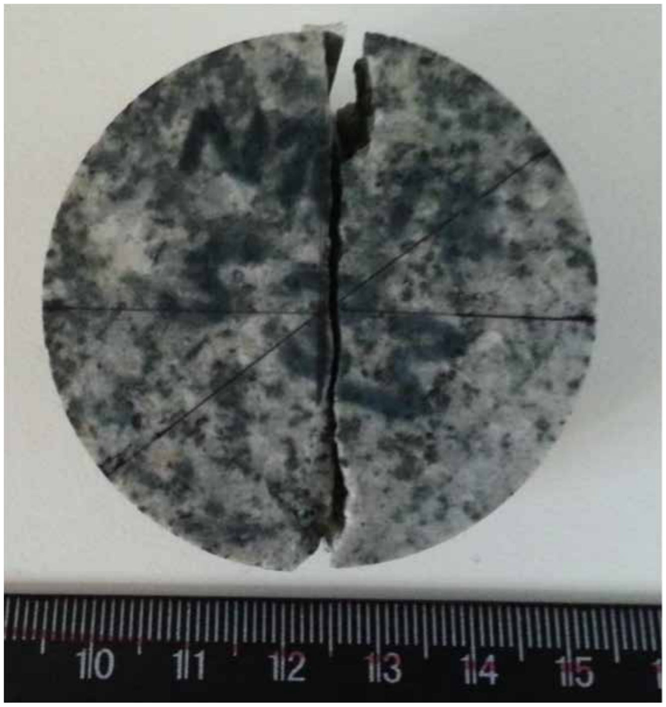

Figure 38: The Brazilian disc sample after the test with tensile failure along the line of loading

3.1.3 BOREHOLE STRESS DETERMINATION METHODS: TELEVIEWER OBSERVATIONS

Context

Borehole breakouts and, more recently, drilling induced borehole wall fractures, are now state-of-the-art methods for indicating stress directions. The shapes, orientations, and spacing of these fractures are often remarkably uniform; and they contain important information on the state of stress. Thus, boreholes are the ideal "stress meter" to assess the stress levels and stress variability within a rock mass. The shapes, too, can provide qualitative information on the relative magnitudes of the in situ principal stresses (i.e. the faulting regime). The spacing of such fractures should be able to provide some indication of stress magnitudes, but this is not yet completely understood. The mapping of such drilling induced fractures during core logging and scanning are a necessary component to any study of stress. Stress concentration results in the greatest shear stress magnitudes at the two opposite azimuths aligned with the least compressive horizontal stress (Sh) (Figure 39).

Spalling of the rock at these azimuths occurs should this stress exceed the material strength; and this results in well-known borehole breakouts. In contrast, tensile drilling induced borehole wall fractures also exist at azimuths orthogonal to the breakouts and provide additional constraints on stress magnitudes (Figure 39).

The two major features related to the in-situ state of stress, borehole breakouts and drilling induced tensile fractures result from concentration of the virgin in situ stresses by the borehole cavity. The interpretation of such features is straightforward when the borehole axis aligns with a principal stress as is the usual case in sedimentary formations with little surface topography. Using this case for illustration, the azimuthal (or hoop) stress (Figure 39) is most compressive at the azimuths perpendicular to the maximum horizontal compression SHmax. Conversely, in the direction of SHmax the hoop stress is more tensile and this can in certain cases produce drilling induced tensile fractures. The azimuths of these features directly and simply indicate the azimuths of the two horizontal principal stresses. One could go further and constrain the stress magnitudes by using the fact that they exist or not (i.e. breakouts but no tensile fractures or vice versa) and the dimensions of the breakouts.

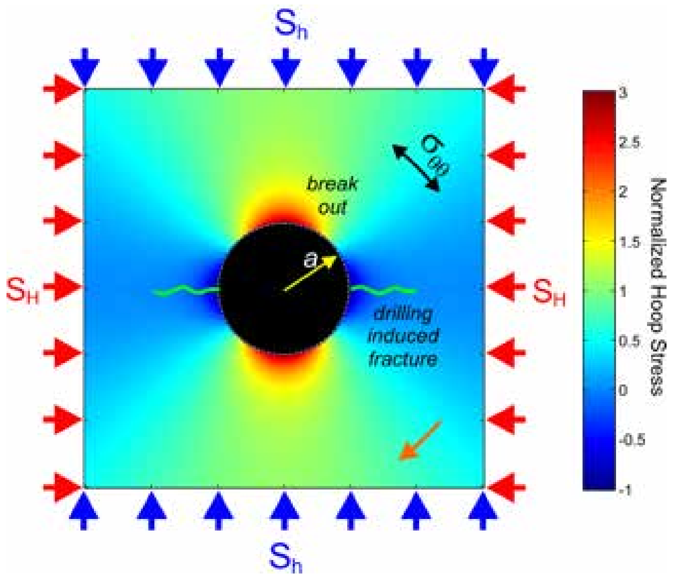

Figure 39: Illustration of the concentration of the azimuthal (hoop) stress by the borehole. Breakouts occur at the azimuth of the minimum horizontal compression S_h. Drilling- induced tensile fractures occur at the azimuth of the maximum horizontal compression S_H.

It is unlikely in the SUMIT study however that the principal stresses conveniently align with the axes of the deviated boreholes A & B (in Figure 35). That being said, both breakout and drilling induced fractures will still be produced but their interpretation is not straightforward. The preliminary discussion below highlights the existence of natural fractures and breakouts. We also tentatively point to evidence for what may be incipient breakouts or spalling on the borehole wall. These 'speckled' zones that appear as parallel bands running along the borehole axis strongly suggest that they are related to the same stress concentrations that produce breakouts.

Such breakouts and drilling induced fractures will be studied using an ultrasonic borehole televiewer (BHTV), and should borehole conditions warrant, optical imaging devices. In operation, the ultrasonic borehole televiewer (BHTV) sends out an ultrasonic pulse (1-2 MHz central frequency) that propagates through the borehole liquid to the borehole wall rock where it is reflected back retracing its path to be recorded. The BHTVs operate with the transducer's beam scanning the borehole wall as the transducer is rotated around the borehole axis while simultaneously being raised up along the borehole. A measure of travel time (TT) and amplitude (A) is obtained about every 1.5° azimuthally and every 5 mm vertically. Travel time and amplitude 'images' provide relatively good spatial resolutions along the borehole axis.

The former TT data can be plotted as a cross section at points along the borehole; this can be used for high resolution caliper measurement or to highlight failed zones. This is particularly useful for the analysis of borehole breakouts although they are most often apparent because of the loss of the reflected signal. Open fractures, too, appear in the image logs because of scattered and lost reflections. Valley et al. (2012) showed that BHTV imaging is fully repeatable in hard rock environments.

Methodology

Here are described some of the features seen from BHTV (ultrasonic borehole televiewer) images obtained in the 1300 and 1700 level boreholes (marked A&B in Figure 35) at the NRS mine.

Continuous BHTV logging was obtained from logger depths of 2.37 m to 389.71 m. An optical televiewer log was also obtained below 300 m depth and it is interesting to first compare the optical to the BHTV images (Figure 41). Note that there are two BHTV images shown in Figure 40. These are the same except that their point of reference changes. In the middle panel the image is oriented with respect to the top (high side) of the borehole and looking down from the surface sweeps clockwise around the borehole azimuth. The rightmost panel shows the same BHTV image but as referenced to the azimuth of magnetic north. Which image is used may depend on the user's needs.

Summary of Findings

The optical televiewer shows in detail much of the geological variability within the formation with light and dark bands representative of the lithology. This is useful in that it can be directly compared to the existing core and used to orient it. It is important to note that a planar feature (such as a natural fracture) that intersects the borehole obliquely will appear in the unwrapped image as a single cycle sine curve. The azimuth of the minimum of the sine directly indicates the dip direction while the sine's amplitude can be converted to dip angle.

The existence of an 'open' natural fracture is readily apparent in the BHTV images by the fact that they make the surface of the borehole rough and scatter the ultrasonic pulse. This causes them to appear as the dark line indicative of lost signal. In contrast, the optical image displays many sinusoidal features that mostly are geological contacts between differing lithologies. Thus, based on this optical image alone it would be very difficult to interpret the sinusoidal feature as a natural fracture as it does not appear significantly different from other sinusoidal features above and below it.

We have also interpreted 'incipient breakouts' in the BHTV images that are dark, speckled bands at opposite azimuths around the borehole. As noted, these may be related to in situ stress but the point here is that while they are very apparent in the BHTV images, this effect is not at all seen in the optical images.

No drilling induced tensile fractures are apparent in the current BHTV images for borehole B. Borehole breakouts however, appear over a number of intervals within B, with one good example provided in Figure 41 which highlights a four-metre section centred at 194 m. Black zones, indicative of lost ultrasonic reflections, are clearly apparent within the unwrapped image in the central panel; such streaks typically are indicative of borehole breakouts in BHTV images. This interpretation is further supported by views of average cross sections (i.e. 2D plots of the measured transit times around a full scan averaged over a short interval) showing the classic 'dog-ear' shape produced by shear failure of the rock at the two opposing azimuths.

The features in Figure 41 are unambiguously interpreted as breakouts on the basis of their azimuthal orientation and symmetry with respect to one another. Plots of the borehole cross-section over these zones show the borehole to primarily be in gauge and circular. Such features are not, to our knowledge, previously described in the literature. Here we tentatively refer to these features as incipient breakouts in which the surface of the borehole appears damaged by, perhaps, irregular spalling over zones subject to high compressive stresses. We believe that these features may provide additional information about the state of stress but we note that this remains to be proven. They are also significantly more prevalent along the borehole than are the classic breakouts.

We further carried out an analysis of the azimuths of both the

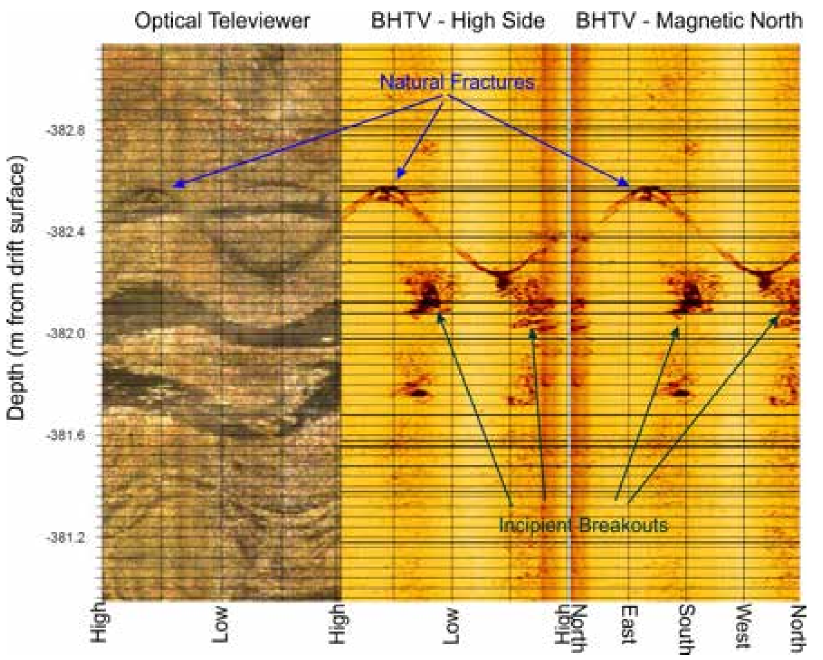

Figure 40: Comparison of optical image log (left panel), borehole televiewer (BHTV) amplitude referenced to high side (middle panel), and BHTV amplitude references to magnetic north (right panel) over the depth range from 380.08 to 382.96 m. (borehole A, Figure 34) Fractures and incipient breakouts are indicated

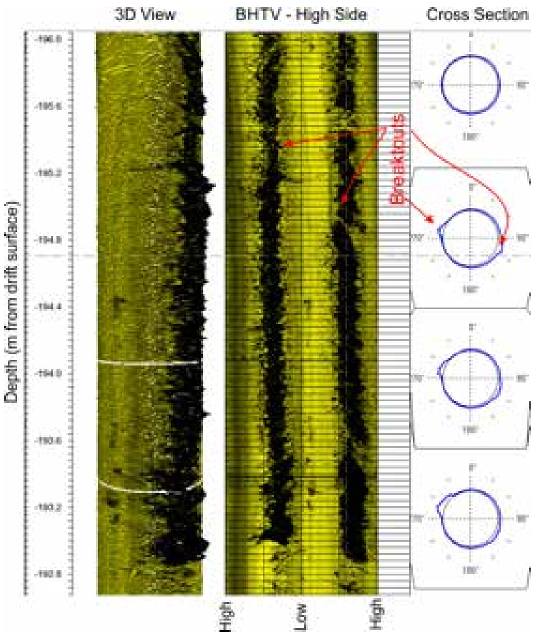

Figure 41: Illustration of classic borehole breakouts from 192 m to 196 m depths in borehole B

observed borehole breakouts and the potential incipient borehole breakouts over the entire length of the borehole. The statistics of the orientations of these features are presented in Figure 42. The breakout directions are remarkably consistent over the entire length of the borehole from depths of 6 m to 391 m. Note however, that Figure 42 is referenced with respect to the high side of the borehole (at 0° such that the vertical direction is indicated by a line from 0° to 180°). One may be tempted to employ the standard interpretation for a vertical borehole aligned with a principal stress, but this is not the case here for the deviated borehole that was arbitrarily aligned with the stress field. A full interpretation of these breakout and incipient breakout directions will require analysis using the more sophisticated full 3D equations that calculate the stress concentrations around an arbitrarily oriented borehole (not shown in this paper).

In contrast, the BTHV data acquired in borehole A show few breakouts or even incipient breakouts. However, there are substantially more fractures observed in borehole A.

The image log for borehole A yielded 525 clearly interpreted fractures (those that could be easily traced as sinusoidal features in the BHTV images) are included. It is important to note that within this data set no distinction was made for whether the fractures might be open (and responsive to the present-day stress field) or if they are closed. One can possibly make this separation by using the corresponding TT log where we might expect an open fracture to scatter the acoustic signal so that it is lost and has a transit time signature. In contrast, during coring the bit passes through a sealed fracture with no obvious expression in the TT image. The orientation statistics of the 525 picked fractures show no single dominant fracture alignments.

Practical Implications:

Borehole televiewer data confirm highly variable breakout and fracture patterns. An analysis that uses only open fractures may also provide additional constraints to the in-situ stress field.

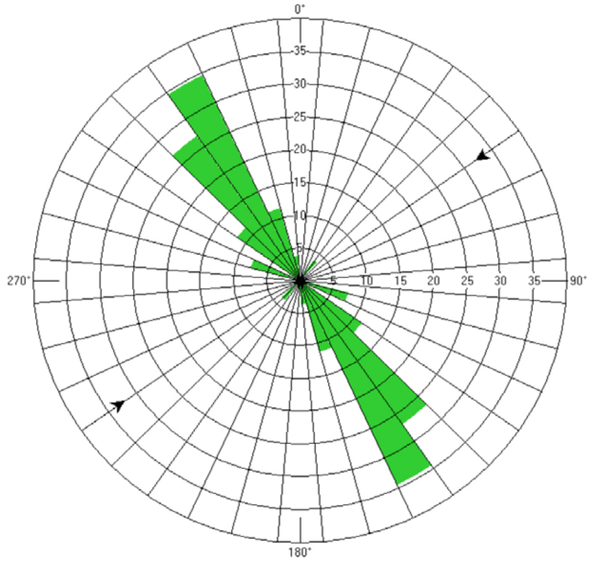

Figure 42: Rose diagram for the orientations of the borehole breakouts and possible incipient borehole breakouts observed in the borehole B BHTV image log. 0° indicates the high side of the borehole

3.1.4 BOREHOLE GEOPHYSICS – FULL WAVEFORM SONIC LOGGING

Context

In many geotechnical applications, borehole logging is used extensively to measure physical rock property contrasts and evaluate the quantitative link between geology and geophysics.

Methodology

Full waveform sonic logging was conducted in two boreholes (marked A & B in Figure 35) using a 1-transmitter and 4-receiver wireline sonic logging tool at the NRS mine. The data was processed using the Full Waveform Sonic (FWS) Processing Module from Advanced Logic Technology's WellCad version 5.0 software package. Raw waveforms were filtered using a frequency bandpass trapezoidal filter to remove noise and capture the acoustic signal detected from the adjacent rock.

In order to obtain velocity measurements from the waveforms, the slowness (inverse velocity) was picked for each waveform along the entire log. Before the slowness is picked, the signals are stacked and cross-correlated between all receivers using the Pro-

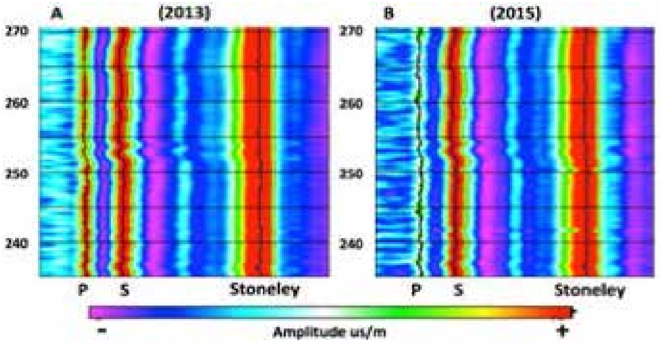

Figure 43: Velocity analysis output for 2013 (A) and 2015 (B) full sonic waveform logs from semblance processing of the filtered and stacked signals for 240-270 m depth of borehole A . Warm colors indicate high correlation of seismic waveform data

cess Semblance algorithm to the semblance at each depth position along the borehole. The semblance is the ratio of coherent energy of the stacked signals to the total energy of the individual signals. An example of semblance outputs from the borehole A is provided in Figure 43.

Summary of Findings

Repeat measurements of Full Waveform Sonic logging were taken from borehole A in 2013 and 2015. It is observed that the waveforms are stable and allow for P-, S-, and Stoneley wave picking from which seismic velocity can be extracted. There were no significant changes from 2013 to 2015. S- and Stoneley waves show no variations in signal. There is a slight amplitude change in P-wave but overall the repeat measurements are comparable.

Once velocity measurements are obtained after semblance processing, they can be used to compute dynamic elastic moduli properties. If elastic wave velocities and the bulk densities are obtained from measurements, then elastic properties for a material could be calculated. However, using high quality waveform data to obtain P-wave (Vp) and S-wave (Vs) seismic velocities, seismic logs can be used to calculate dynamic elastic properties (Yilmaz, 2015; Schoen, 1996). Young's modulus relates to the amount of stiffness of an elastic solid expressed by the ratio of stress to strain in a uniaxial state.

Using Vp and Vs measurements from borehole B and density bands of 2.7, 2.8, 2.9 g/cc, a pseudo-Young's modulus is calculated. The velocity analysis and Young's modulus measurements for borehole B are provided in Figure 44. Additionally, several deformational lab experiments were conducted on samples from borehole B measuring the tensile strength of the rock. It can be observed that the lab samples and pseudo Young's modulus follow the same trends along the borehole changing in parallel over different lithological units relating to the different stiffness of each rock type. Similar trends are observed in data derived from uniaxial compressive strength (UCS) and Brazilian disc test for tensile strength (Figure 38).

The measured variation in the rock properties (such as P- an S-wave velocities, dynamic Young's modulus, uniaxial compressive strength, and tensile strength with depth) are plotted in Figure 44 and show the variation of geophysical parameter and elastic moduli tensile with depth and geology. Figure 44 also shows that there are significant changes in these elastic as well as strength properties due to change in the lithology along the depth of both the cores. Horizontal bars indicate the results from the loading measurements for the borehole core samples (from 0 MPa to 50 MPa, Figure 37) compared to the corresponding in situ borehole velocities and derived elastic moduli. At 50 MPa, the lab measurements and the in-situ borehole data are in good agreement. Also, note zones of high and low dynamic Young's modulus exist in borehole B.

Practical Implications

Full waveform sonic data show highly variable velocities and associated dynamic elastic moduli. The combined lab and field studies confirm that full waveform sonic data can be used to obtain reliable estimates of in situ dynamic elastic moduli (at elevated in situ stress levels > 35 MPa).

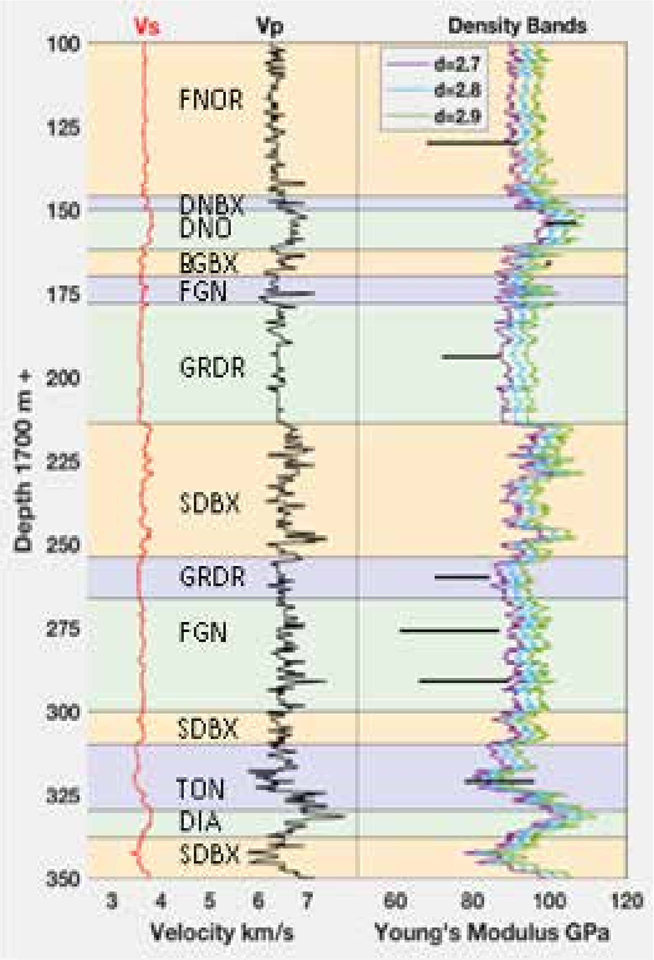

Figure 44: Compressional (P) and shear (S) wave velocity logs in borehole B combined with dynamic Young's modulus estimates from lab measurements and from full wave form sonic data. At 50 MPa, the lab measurements and the in situ borehole data are in good agreement. Also note, zones of high and low dynamic Young's modulus exist in borehole B

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NOTE: For a complete listing of the authros' publications please go to:
https://scholar.google.ca/scholar?q=bernd+milkereit&hl=en for Milkereit
https://scholar.google.ca/scholar?q=kaiwen+Xia&btnG=&hl=en&as_sdt=0%2C5 for Xia and
https://scholar.google.ca/citations?user=fMsSirgAAAAJ&hl=en for Young

Acknowledgements

The University of Toronto projects were funded as a SUMIT ORF project coordinated by CEMI with additional graduate student support from NSERC. We would like to thank Sudbury Integrated Nickel Operations, a Glencore Company for providing access to the NRS Mine site. Sudbury Integrated Nickel Operations, a Glencore Company provided operational support for in-mine surveys, provided geological and geotechnical data for the 3D/4D mine model, and provided dedicated boreholes for geophysical, petrophysical and time-lapse geophysical studies. The integration of time-stamped infrastructure and production data (stopes) into the 3D geological GoCad mine model was conducted by Mira Geoscience.

3.2 Towards Far-Field Stress Calibration Through Measuring Tunnel Deformation with LiDAR Imaging

Team:

Queen's University – *Dr. Steve McKinnon* and *Connor McAnuff* and, Carleton University – *Dr. Claire Samson.*

Title: Dr. Steve McKinnon, Professor and Chair in Mine Design
Role: Co-PI
Collaborators on team: Carleton University, Sudbury Integrated Nickel Operations – A Glencore Company

Goals

Document a procedure for accurately measuring excavation-induced drill and blast tunnel deformation using LiDAR point clouds.

Develop a method to estimate the orientation and magnitude of excavation-induced stress changes though back-analysis of tunnel deformation measurements for use in far-field stress calibration.

Abstract

The practical feasibility, applications, and limitations of calibrating far-field stress estimates through tunnel deformation measurements captured using LiDAR imaging have been investigated. LiDAR point clouds currently have many underground mining applications, so it is desired to determine additional uses for the data. A method, based on the superposition of stresses, that produces estimates of the orientation and magnitude of excavation-induced stress changes through back-analysis of deformation measurements from LiDAR imaged tunnels has been developed and tested using synthetic data. Excavation-induced stress estimates can be used for far-field stress calibration.

Initial testing of the method was done under ideal conditions, with various scenarios representing more realistic conditions tested separately. Under ideal conditions, the back-analysis method estimated stress change orientations within ±5°, and magnitudes within ±2 MPa. Preliminary testing involving plastic deformation, rough tunnel profiles, and profile occlusions suggests that the method can work under more realistic conditions. In addition to the back-analysis estimation method, a procedure based on existing techniques to accurately measure tunnel deformation using LiDAR imaging has been documented.

Context

Traditional methods of far-field stress estimation such as overcoring, hydraulic fracturing, and flat-jack methods are single-point measurements of stress and can be a poor representation of far-field stresses, as point measurements can be influenced by nearby excavations and geological structures. These methods are expensive and time consuming, which reduces incentive to perform supplementary tests. For these reasons, far-field stress estimates can be associated with a high level of uncertainty. It is desired to reduce stress estimate uncertainty to improve mine design through a better understanding of the surrounding rock mass.

Methodology

One method of reducing uncertainty is through the calibration of numerical models by matching model output to field observations. It is proposed that far-field stress estimates are calibrated by minimizing differences between theoretical excavation-induced stress changes output by a numerical model and excavation-induced stress changes estimated through back-analysis of tunnel deformation measurements. A 2D numerical model of the deforming tunnel is required for back-analysis. The model is used to define calibration curves by applying stress changes at known orientations and magnitudes and calculating the resulting deformation. Deformation measurements of the modelled tunnel profile are then compared with the numerical calibration curves to estimate the orientation and magnitude of the principal stress change causing tunnel profile deformation. Estimation of stress change orientation and magnitude is founded on the superposition of stresses in a linearly elastic medium and performed by minimizing squared differences between calculated tunnel and calibration curve deformation.

A total of 84 back-analysis tests using numerical models have been performed to assess the method's proficiency for a linearly elastic medium. Four different tunnel profile shapes were used (Figure 45) to show the method is not dependent on a certain profile shape. In deep mines, it is likely there will be some degree of fracturing and plastic deformation around excavations.

Tests have been performed on a linearly elastic medium to isolate and analyze the effects of various scenarios. For each model, uniaxial, biaxial, and hydrostatic stress changes 2.5 to 30 MPa in magnitude were applied at various orientations. For base level tests, perfect knowledge of the rock mass properties was assumed. Following base level testing, more realistic scenarios were tested including a distorted tunnel profile, profile occlusions, inaccurate rock mass properties, and plastic deformation (Figure 46).

Summary of Findings

The method was accurate in estimating the orientation and magnitude of various stress changes under ideal conditions. The maximum orientation estimate deviation was 5°, with the majority of estimates having no deviation. Magnitude estimate deviations averaged less than 0.5 MPa with a maximum deviation below 2 MPa. Hydrostatic stress changes resulted in the greatest estimate deviations. Sensitivity analysis suggests that a distorted tunnel profile and occlusions have little effect on estimates. Plastic deformation had varying effects on stress change magnitude estimates and no effect on stress change orientation estimates.

Practical Implications

The results from this project set the groundwork for the continued development of a far-field stress estimate calibration method using tunnel deformation measurements. The next step would be to test the method on a real tunnel, measuring deformation using the proposed procedure. Further theoretical testing would be beneficial for method refinement.

LiDAR point clouds have been shown to have underground applications such as shotcrete thickness calculations, identifying regions of potential leakage, large-scale roughness determination, detecting geological features, and as-built bolt spacing measurements. With LiDAR data being collected for these purposes, a method of far-field stress calibration using point clouds would be an inexpensive and efficient method to improve on far-field stress estimates for mine design purposes.

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NOTE: for a complete listing of the author's publications please go to:
http://my.mine.queensu.ca/People/StephenMcKinnon.html#f-ndtn-publications

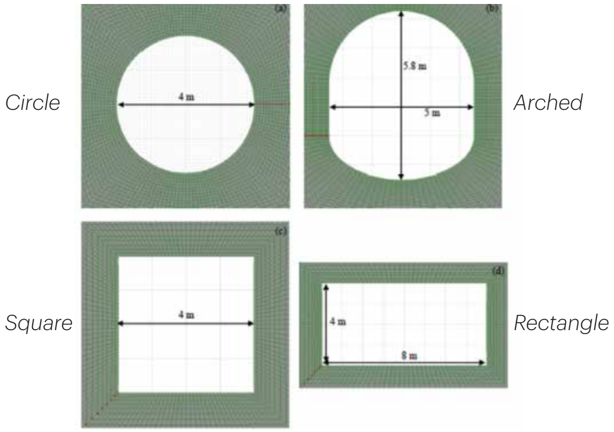
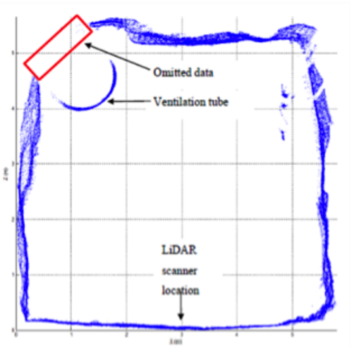
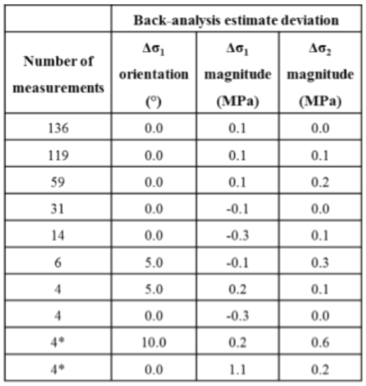


Figure 45: Tunnel shapes selected to show method is independent of profile selected





	Back-analysis estimate deviation		
Number of measurements	$\Delta\sigma_1$ orientation (°)	$\Delta\sigma_1$ magnitude (MPa)	$\Delta\sigma_2$ magnitude (MPa)
136	0.0	0.1	0.0
119	0.0	0.1	0.1
59	0.0	0.1	0.2
31	0.0	-0.1	0.0
14	0.0	-0.3	0.1
6	5.0	-0.1	0.3
4	5.0	0.2	0.1
4	0.0	-0.3	0.0
4*	10.0	0.2	0.6
4*	0.0	1.1	0.2

Figure 46: Realistic scenarios including tunnel distortions and occlusions were examined

3.3 Real-Time Monitoring of Geophysical Data in Deep Mines

Team:
University of Toronto (see 3.1)

Project Goals

As physical rock properties, such as compressional wave velocities and conductivities are sensitive to stress changes, continuous monitoring of 3D controlled source geophysical data may provide new insights into spatial and temporal variations of stress at the SUMIT test site.

Abstract

Over a two-year period, the large volume rock mass (of approx. 2,000,000 m^3) in the immediate vicinity of an active mining zone exhibited reduction in resistivity, indicating small fracture porosity reduction due to higher stress levels. In contrast, a large rock mass surrounding a much deeper control borehole away from mining showed no change in physical rock properties over the same period of time. Borehole-based electrical measurements are ideally suited to monitor the rock volume around a borehole as resistivities are more sensitive to temporal stress variations than seismic velocities (for very low fracture porosities).

3.3.1 MONITORING CHANGE – TIME-LAPSE GEOPHYSICS

Context

For earth materials within lithostatic pressure, the bulk electrical resistivity of the materials is not only controlled by the conductive phases, but largely influenced by combinations of porosity, permittivity, fracturing, fluid content and their interactions (Archie, 1942, Ward, 1990, Glover, 2010). Under various experimental settings, researchers have reported a relationship between resistivity of crystalline rocks and compressional stress and such a relationship can be highly non-linear. In general, stress induced change of resistivity in crystalline rocks can be attributed to changes in pore shape and pore space, which closes or opens cracks and fractures and alters the conduits of fluid and current flow within the rock volume.

The methods detailed in the earlier reported work (Borehole logging and petrophysical lab studies) currently represent the state of the art with respect to stress determination from deeper ‘slim-hole’ wellbores. Other technologies do exist to measure stress, but at least with current availability, these require larger wellbores more typical of the petroleum industry (e.g. shear wave anisotropy from dipole sonic logging, high resolution mechanical caliper measurements) or are only appropriate in shallow borings extending from mines (e.g. over-coring methods, optical holography). A disadvantage of these methods is that they can usually only be applied once (drilling induced fractures) or due to the expenses of logging and hydraulic fracturing they can only be applied a limited number of times.

Although this has not to our knowledge been attempted, use of both borehole imaging and even hydraulic fracturing in a time-lapse sense to look for potential changes with time will be one possible option for observing changes in the stress. A repeated survey could be employed, for example, in the time period follow-

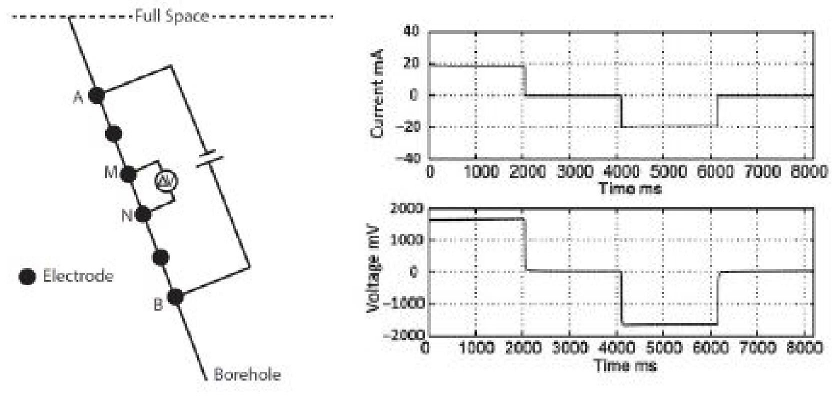


Figure 47: Direct Current Resistivity/Induced Polarization measurement acquisition geometry along a borehole in full space. Current is injected at electrode A (+I) and sinks at electrode B (-I). Potential difference ΔV is measured between electrodes M and N. b). Current and voltage waveform example

ing a significant microseismic event within the volume of rock to be studied, although there could be logistical difficulties with this approach.

Methodology

Over the course of 2013 to 2015, time-lapse direct current resistivity and induced polarization (DC/IP) surveys were conducted along two boreholes as part of the multi-parameter geophysical surveys in a deep highly stressed mine located in Sudbury, Ontario, Canada. The two boreholes are approximately 400 m long with NQ diameters at depths of about 1300 – 1600 m and 1700 – 2000 respectively (Figure 35). The DC/IP measurements were taken using a Wenner array with minimum electrode spacing of 4 m (Figure 47).

Current is injected at electrode A (+I) and sinks at electrode B (-I). Potential difference ΔV is measured between electrodes M and N in the centre of current electrode A and B. During the survey, the current electrode spacing AB is increased while the potential electrode spacing MN remains the same. The electrode array AMNB is moved along the borehole to cover the entire borehole profile. The potential difference between M and N

$$\Delta V = \rho_a I / 4\pi \left(\frac{1}{AM} - \frac{1}{BM} - \frac{1}{AN} + \frac{1}{BN}\right) \quad (1)$$

$$\rho_a = \frac{\Delta V}{I} k, k = 4\pi \Big/ \left(\frac{1}{AM} - \frac{1}{BM} - \frac{1}{AN} + \frac{1}{BN}\right) \quad (2)$$

and apparent resistivity (ρa) can be calculated by rearranging Equation (1) where k is the geometry factor for a certain electrode array. The apparent resistivity can be plotted against N-spacing of the electrode array and gives the apparent resistivity pseudo-section, in which N is multiplier of the minimum electrode spacing as spacing between current electrodes A and B are increased during the survey. In general, greater N corresponds to greater depth (or radius, in the case of a borehole) of investigation. A large, multi-electrode DC/IP borehole array will sample a large volume rock mass (of approx. 2,000,000 m^3 for N=10) around the borehole.

Valley et al (2012), showed that the multi-electrode borehole DC/IP acquisition system is able to obtain reliable down-hole measurements and provides excellent repeatability that favours 4D monitoring surveys.

Summary of Findings

Figure 48 (a) to (c) depict the time-lapse apparent resistivity pseudo-sections of borehole A (1300 level) from 2013 to 2015. N-spacing indicated radius around borehole A (N=1 represents 4 m radius, N =10 represents 40 m radius). The red colours at large radii (9<N<11) indicates the presence of the conductive ore body at about 40 m distance. Note the overall decrease in resistivity over the two year-period. Comparing resistivity data shown in (a), (b), the off-hole apparent resistivity along the 1300 level borehole increased up to 400%. Correspondingly, between 2013 and 2014, the 1300 level was experiencing active mining operations nearby.

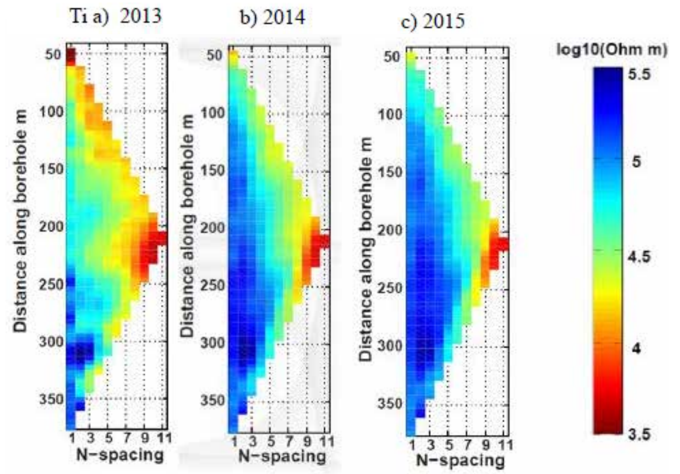


Figure 48: Apparent resistivity pseudo-section of 1300 level borehole measured in 2013 (a), 2014 (b), 2015(c). The resistivity section samples of 3D volume around borehole A. N spacing indicated radius around borehole A (N=1 represents 4 m radius, N =10 represents 40 m radius). Note the overall decrease in resistivity over the 2-year period. The red colour at large radii (9<N<11) indicates the presence of the conductive ore body at about 40 m distance

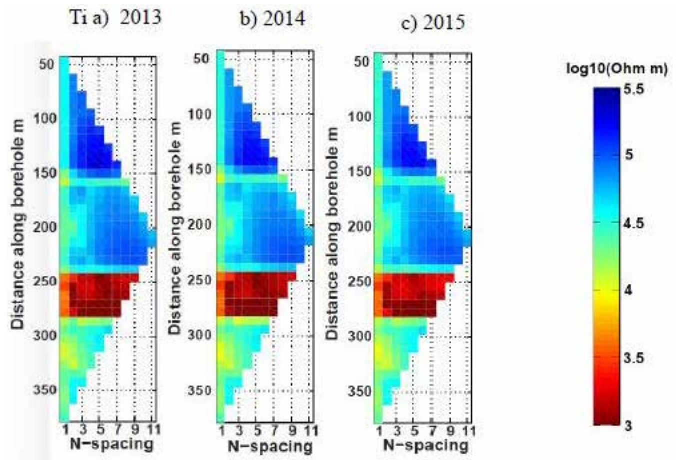


Figure 49: Apparent resistivity pseudo-section of 1700 level borehole B measured in 2013 (a), 2014 (b) and 2015 (c). The section samples a volume around borehole B and no significant change in bulk resistivity are observed at the deeper exploration area

The full-waveform sonic tool, which has a much higher lateral resolution but shallower depth of investigation, captured almost no difference. Our petrophysical analysis shows the rock mass at the mine is mostly resistive and has less than 1% porosity (Figure 35). The recorded increase in resistivity may be related to an increase in the stress field due to nearby rock extraction, which closes micro-cracks and reduces the total porosity of the rock mass. Such a small change in porosity (less than 1%) does not result in observable change in acoustic velocity. However, it can increase the electrical resistivity by orders of magnitude in a resistive environment.

On the deeper 1700 level (borehole B in Figure 35), the large array borehole resistivity data were acquired in 2013 (a), 2014 (b) and 2015 (c). The resistivity pseudo section sampled a volume around borehole B with no significant change in bulk resistivity being observed (Figure 49).

At the deeper exploration level, the rock mass only experiences exploration activities such as exploration drilling and not much change in stress field is expected. Correspondingly, our resistivity measurements show no time lapse differences (compare results with time-lapse data from the 1300 level shown in Figure 48).

Practical Implications

The non-destructive borehole direct current (DC) electrical resistivity measurements are sensitive to small change in pore space in a resistive environment. This favors the use of borehole resistivity tools for time lapse stress monitoring of a large volume, low porosity rock volume in an active mining environment.

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3.3.2 H/V SEISMIC RATIOS FOR MONITORING CHANGE

Context

Exploiting the observed dependencies of seismic velocity to stress and fracturing, it is possible that in situ passive seismic monitoring in deep mine settings can be used to measure the evolving stress regime and/or rock property changes. The intent is to develop a real-time method that locally samples an immediate rockmass to detect any changes that could lead to failure (i.e. a proactive mine-safety device). This contrasts with currently utilized microseismic systems that detect activity and calculate the event location to identify instability (a reactive system). A very simple and attractive method is comparing the horizontal and vertical seismic energies at a location, the H/V method.

The H/V method works on isolated 3-component receiver sites and utilizes the horizontal to vertical component transfer functions. Nakamura (1989) pioneered this on the assumption that the horizontal component primarily consists of the S-wave whereas the vertical is due to the P-wave. Simply put, the H/V ratio is a measure of S- to P-wave energy partitioning. Increased signal in the horizontal component indicates greater S-wave amplification, and thus higher risk to structures at surface. Building on this method, inverting of the H/V spectral data can produce S-wave velocity models (Arai and Tokimatsu 2004).

Also, as shown in Figure 50, body waves incident on a free surface show partitioning that is dependent on the Vs to Vp ratio (or Poisson's ratio) (Miller and Pursey 1954). The problem is how to isolate just the body wave energy where surface waves dominate.

In an underground setting, the tunnel infrastructure does provide a limited free surface to enact energy partitioning. Tunnel seismology studies show that the mode of wave propagation changes as the wavelength approaches that of the tunnel diameter (Jetschny et al. 2010). For shorter wavelengths, the tunnel wall appears much like that of a free surface over a half space, producing

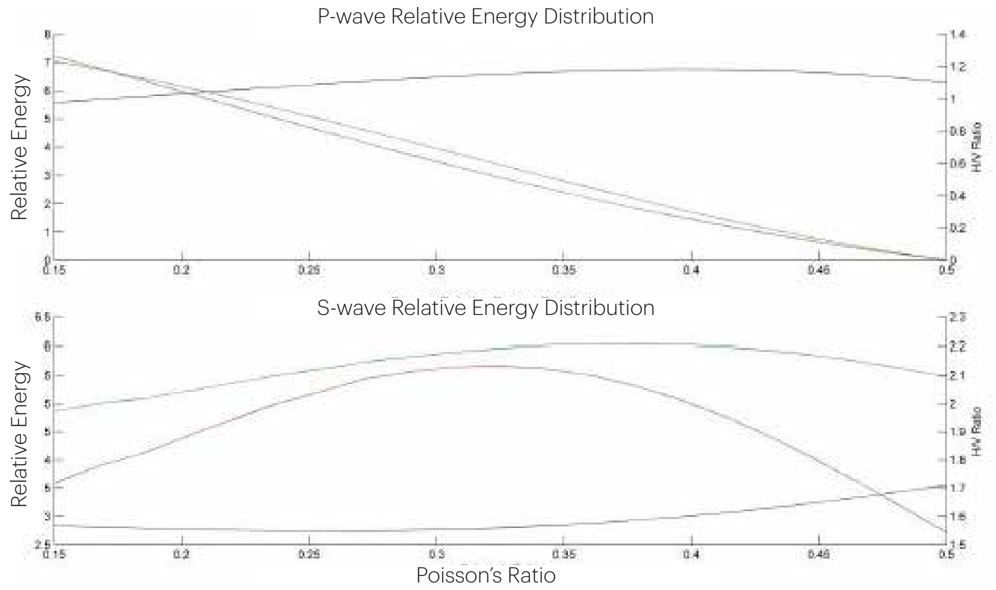

Figure 50: For 3-component seismic sensor Integrating the relative energies over all incident angles, the vertical energy component (blue) is less affected by variation in Poisson's ratio than that of the horizontal (green) for both incident P-waves (top) and S-waves (bottom). It is then suggested that a change in the ratio of the energy components (red) could be an indicator of changing rock properties in immediate vicinity to the 3-component sensor

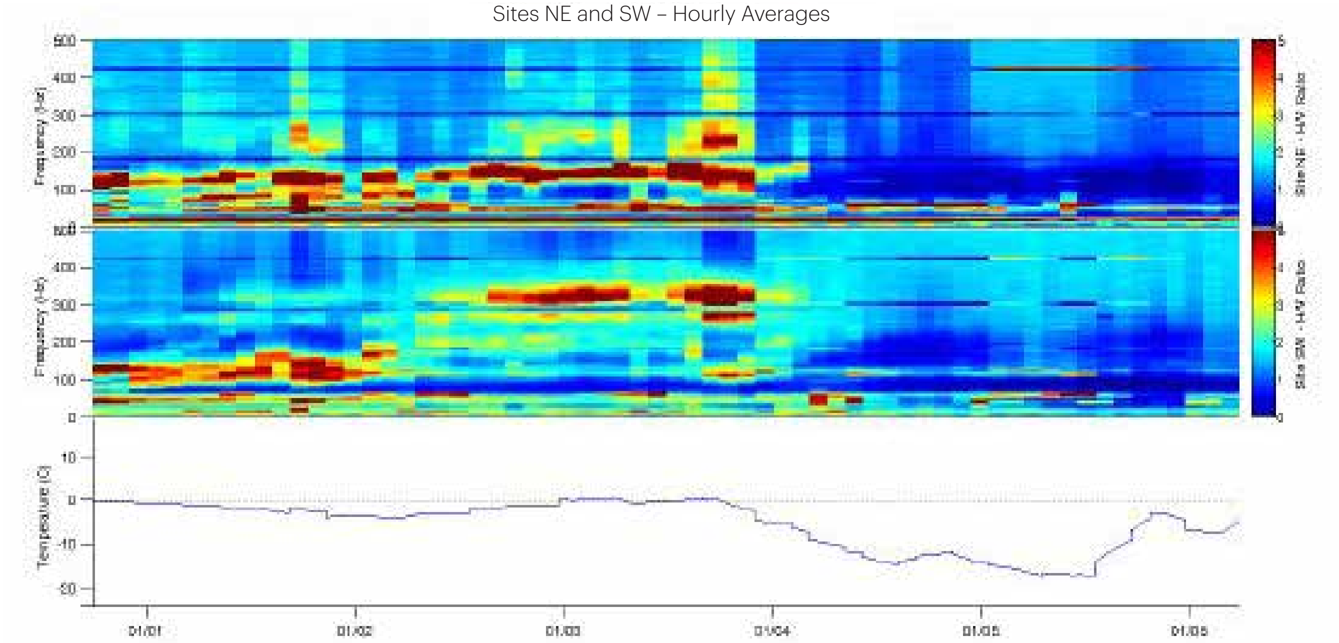

Figure 51: Proof of concept: hourly H/V spectral ratio averages of two sites (top two strips) over several days during a freeze cycle (temperature in lower strip). Resonances at 100Hz and 200Hz disappear as the ground freezes at both sites. When thawing occurs, they are re-established

surface waves. These are similar to Rayleigh waves, except in that they wrap around the tunnel. At wavelengths greater than the tunnel diameter, particle motion tends towards body wave motion. For typical hard rock mines, these wavelengths translate to frequencies in excess of 1kHz.

Methodology

On the assumption that stable H/V ratios can be generated in a mine environment, the sensitivity, time-averaged stability and resolution need to be sufficient to show change over longer periods in response to changing rock-mass properties. An experiment monitoring two 3-component sites over a winter was performed to determine if freeze/thaw cycles could be observed.

Summary of Findings

Figure 51 shows a freezing cycle in which H/V ratios reduce with large amplitude resonances disappearing. After thawing, responses are re-established. This experiment shows that: stable H/V estimates are generated and do change predictably with ground conditions.

Practical Implications

It may be possible that in situ passive seismic monitoring in deep mine settings can be used to measure the evolving stress regime and/or rock property changes and that a real-time method that locally samples an immediate rock-mass to detect any changes that could lead to failure (i.e. a proactive mine-safety device) can be developed.

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NOTE: for a complete listing of the author's publications please go to:
https://scholar.google.ca/scholar?hl=en&q=bernd+milkereit

3.4 Remote Geophysical Monitoring of Stress/Strain

Team:

Laurentian University – *Dr. Richard Smith* and *Christoph Schaub*

Title: Dr. Richard Smith
Role: Co-PI
Collaborators on team: Vale

Project Goals

The initial intent of this sub-project was to see if there were impulsive random events associated with strain events that were also evident in the data collected.

Abstract

A test survey in a parking lot indicated that rocks subjected to stress showed signs of strain (cracking) that could be detected remotely using a magnetic field sensor. The cracking also generated an audio signal. Further work would be required to test this in a mine using sensors that are cheap enough to be deployed close to areas that might be subject to stress and strain.

Context

Monitoring stress and strain is important in a mine environment as stresses can lead to potentially damaging rock bursts. Recent work claims that stress in rocks results in measurable random electric field emission (Lichtenberger, 2006; Greiling & Obermeyer, 2010; Krumbholz, 2010.

Methodology

An experiment was designed to see of the magnetic field emissions associated with strain related events could be detected. The survey was undertaken in a parking lot, distal from sources of anthropogenic magnetic noise. The sensor was an extremely sensitive magnetometer designed for measuring changes in the magnetic field associated with fluctuations in the ionosphere and lightning strikes in tropical regions. Thirty-two rock samples were placed under increasing axial stress in a load frame. The magnetic field was measured as the pressure increased, and random events were seen in the data.

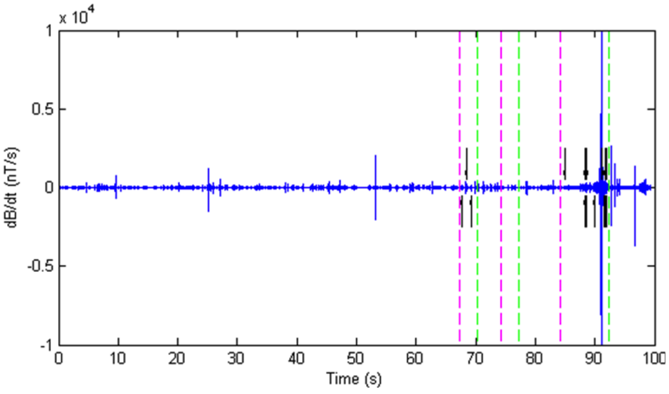

Figure 52: Time derivative of the magnetic field shown in blue. Magenta lines indicate the onset of audible cracking; green lines indicate their cessation. The final green line also indicates sample failure. Black arrows indicate magnetic emissions believed to be associated with stress events

Summary of Findings

It was observed that acoustic (audio) emissions (AEs) were heard as the rock was stressed, particularly prior to failure. For some of the samples, a video was taken of the process. Subsequent analysis of the magnetic field data and the AEs showed that distinct emission events in the magnetic field data corresponded to AEs heard on the video.

Figure 52 shows the time derivative of the magnetic field as a function of time. The time derivative amplifies the high-frequency emissions and suppresses low-frequency variations.

Practical Implications

Magnetic field sensors or acoustic sensors could be used to detect strain events in rocks that are stressed. The experiment was undertaken in a low-noise environment. If this procedure is to be successful in a mine, then further testing and development work is required. Firstly, the experiment has to be repeated in a high-noise environment and a procedure for identifying and differentiating strain related emissions from anthropogenic events has to be developed. Secondly, cheap sensors have to be designed that would allow many sensors to be placed in a number of locations within a few metres of where strain events are expected or

References Cited

Greiling, R. O., and H. Obermeyer, 2010, Natural electromagnetic radiation (EMR) and its application in structural geology and neotectonics: Journal Geological Society of India, 75, p.278–288.

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Lichtenberger, M., 2006, Underground measurements of electromagnetic radiation related to stress-induced fractures in the Odenwald Mountains (Germany): Pure Applied Geophysics, 163, p.1661–1677.

NOTE: for a complete listing of the author's publications please go to: https://scholar.google.ca/citations?user=Yvv8uccAAAAJ&hl=en&oi=ao

3.5 Dynamic Support Research for More Effective Mine Design

Team:
Laurentian University – *Dr. Ming Cai* and *Xin Wang*

Title: Dr. Ming Cai, Professor,
Geomechanics Research Chair
Role: Co-PI
Collaborators on team: Vale

Project Goals

This subproject aims at contributing to improving our understanding of seismic wave propagation in deep underground mines with a particular emphasis on ground motion near excavation boundaries due to seismic wave propagation that resulted from a remote fault-slip seismic event. Advanced numerical modeling tools will be used and new modeling techniques will be developed to attain this goal.

Abstract

Rockburst damage localization, which is not well understood, has been observed in deep underground mines. Analysis of seismic wave propagation in underground mines is of great interest for improved understanding of the dynamic rock failure problem, which is important for improved rock support design.

In this subproject, research is emphasized on ground motion around excavations due to seismic wave propagation that resulted from a fault-slip seismic event in deep mines. It is found that: (1) a point source model (i.e., moment tensor model) and a non-point source model (i.e., kinematic rupture model) are suitable for the far-field and near-field source representations, respectively; (2) ground motion is influenced by many factors such as target-source distance, slip direction, spatial location, and geometrical and geological conditions; (3) PPV (peak particle velocity) values increase as the wavelength-to-excavation span ratio increases and ground motion amplification increases as this ratio decreases; (4) shear wave quality factor can be estimated by comparing modeling results with that from a scaling law; (5) wave propagation patterns around an excavation can be altered and PPV amplification and shielding effect can occur near the excavation boundaries due to heterogeneities such as tunnels, open and backfilled stopes, and dykes in deep mines; (6) a coupled numerical method to construct non-uniform velocity models is proposed.

Relatively stronger amplification is observed in the low confinement zones and on the excavation surface in the non-uniform velocity models. The proposed coupled numerical procedure offers a method to improve the understanding of the site amplification effect and ground motion near excavation boundaries; (7) non-point source modes could be a better representation of the fault-slip source than the point source model for near-field sources.

Context

Mining-induced seismicity and rockbursts increase continually and cannot be prevented as mining progresses to deeper levels. It is often observed that after a large seismic event, damage is often localized. In other words, the damage extent along a tunnel varies for the same seismic event. Damage does not necessarily occur only at locations nearest to the seismic source. It has been reported that damage can occur at locations quite far from the seismic source and in unexpected areas. It is possible that tunnels located at a large distance from the seismic source may suffer more severe damage, while tunnels near the seismic source may experience less damage, as illustrated in Figure 53. This phenomenon, in turn, reveals a need for improved dynamic support design.

Dynamic ground support design against large seismic events usually involves uncertainty in design input parameters such as PPV (Peak Particle Velocity). Many factors such as excavation shape and size, wave attenuation and geological structures can lead to the uncertainty of PPV. It is important to study the influence of these factors on ground motions due to seismic wave propagation from a fault-slip seismic event. Hence, an improved understanding of seismic wave propagation in a complex mining context is essential for a better understanding a rockburst damage localization and for improved rock support design in burst-prone mines.

In deep mines, both analytical and experimental methods are of limited use in solving seismic wave propagation problems due to complex geological, geometrical, and stress conditions. Hence, ground motions near excavation boundaries in underground mines are currently not well understood. Fortunately, with the

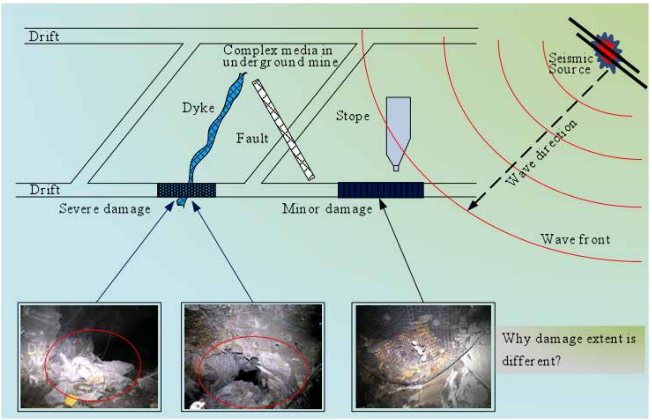

Figure 53: A sketch to illustrate different degrees of rockburst damage at various locations in a mine

rapid advancement of computer technology and many powerful numerical techniques, numerical modeling is becoming an irreplaceable tool. In this project, SPECFEM2D/3D codes are employed to model seismic wave propagation.

Methodology

As shown in Figure 54, six main research components are addressed in this project.

Numerical modeling is the main approach for this project. Different numerical steps along with the selected numerical tools are shown in Figure 55. In the pre-processing step, several software packages are used to generate meshes. Matlab, ParaView, and Origin are used for post-processing of results. Based on the merits of FLAC and SPECFEM codes, we developed a new coupled approach to model a non-uniform velocity field. It provides a new way to study the ground motion near excavation boundaries.

Summary of Findings

(1) Advanced numerical codes SPECFEM2D and SPECFEM3D are robust tools that can be used to conduct seismic wave propagation modeling in deep mines. Ground motion depends not only on the target-source distance, but also on factors that affect wave patterns. Seismic response around excavations is strongly wave pattern-dependent.

(2) Wave fields become more complex as the ratio of wavelength-to-excavation span decreases. PPV values around an excavation increases as the ratio increases. The amplification factor is independent of the ratio when the ratio is relatively large.

(3) Wave propagation patterns are altered largely by various heterogeneities in deep mines. Both PPV amplification and shielding effects can occur near excavation boundaries.

(4) A nonlinear velocity model, which links wave velocity to confinement and rock mass quality, is presented. A FLAC/SPECFEM2D coupled approach is developed based on the proposed nonlinear velocity model.

(5) Confinement change and rock mass failure can alter the velocity field around an opening. Strong amplification normally occurs in the low confinement zones and on the

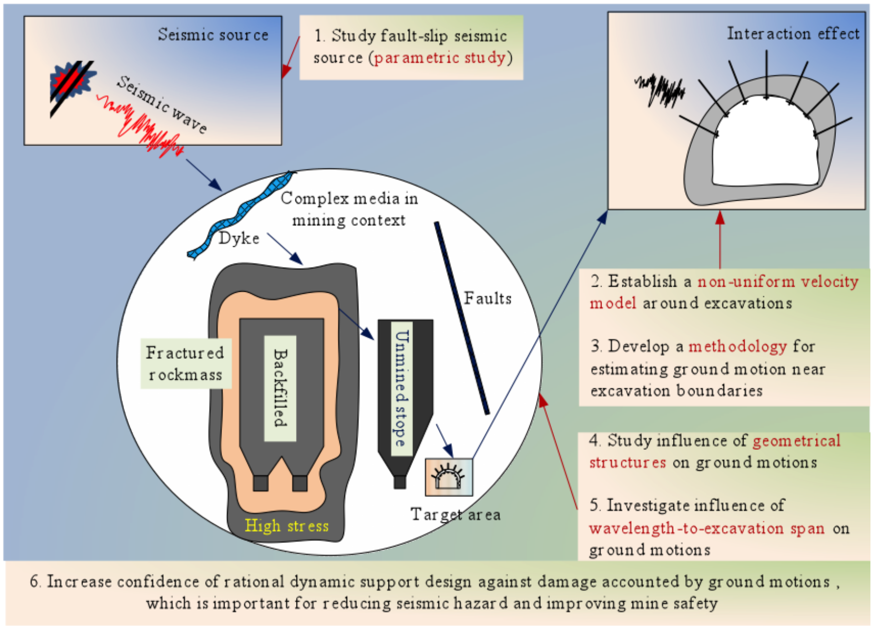

Figure 54: A sketch illustrating the main research components in this subproject

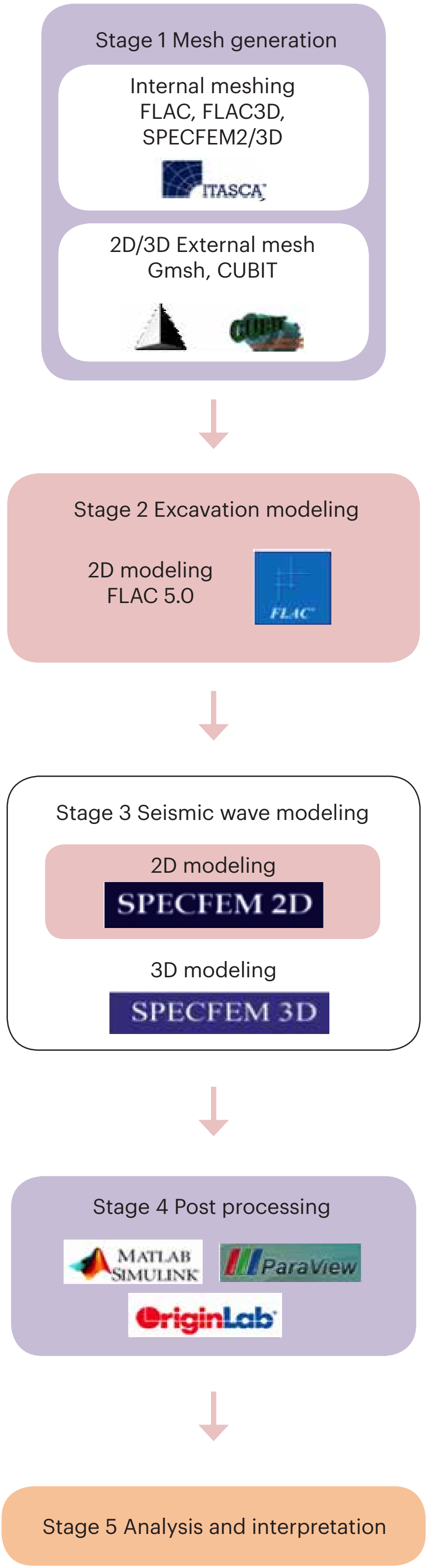

Figure 55: Main numerical steps and tools employed in the subproject

excavation surface. Complex wave fields are also observed in the non-uniform velocity models.

(6) By combining SPECFEM3D and a non-point source model it is possible to capture realistic ground motion due to seismic wave propagation in underground mines (Figure 56). This in turn can assist in rock support design and rockburst risk management.

Practical Implications

- A better understanding of seismic wave propagation in underground mines is achieved. The influence of typical heterogeneities (i.e., tunnels, open and backfilled stopes, and dykes) on ground motion has been investigated. Figure 57 presents snapshots of horizontal velocity components at four propagation times, showing the influence of a dyke, a stope, and a tunnel on the wave field.
- A new method to estimate shear quality factor for hard rocks in deep mines is proposed.
- A new confinement and rock quality dependent non-linear velocity model is developed.
- A new coupled FLAC/SPECFEM2D modeling technique for seismic wave propagation simulation is developed. As shown in Figure 58, different wave fields are observed for uniform and non-uniform velocity models.
- The method and models developed in this study can be used for forensic study of rockburst damage in deep mines. This in turn can help government and mining companies to make decisions to further improve mine safety and mining companies to make decisions to further improve mine safety.
- A better microseismic source location determination can be

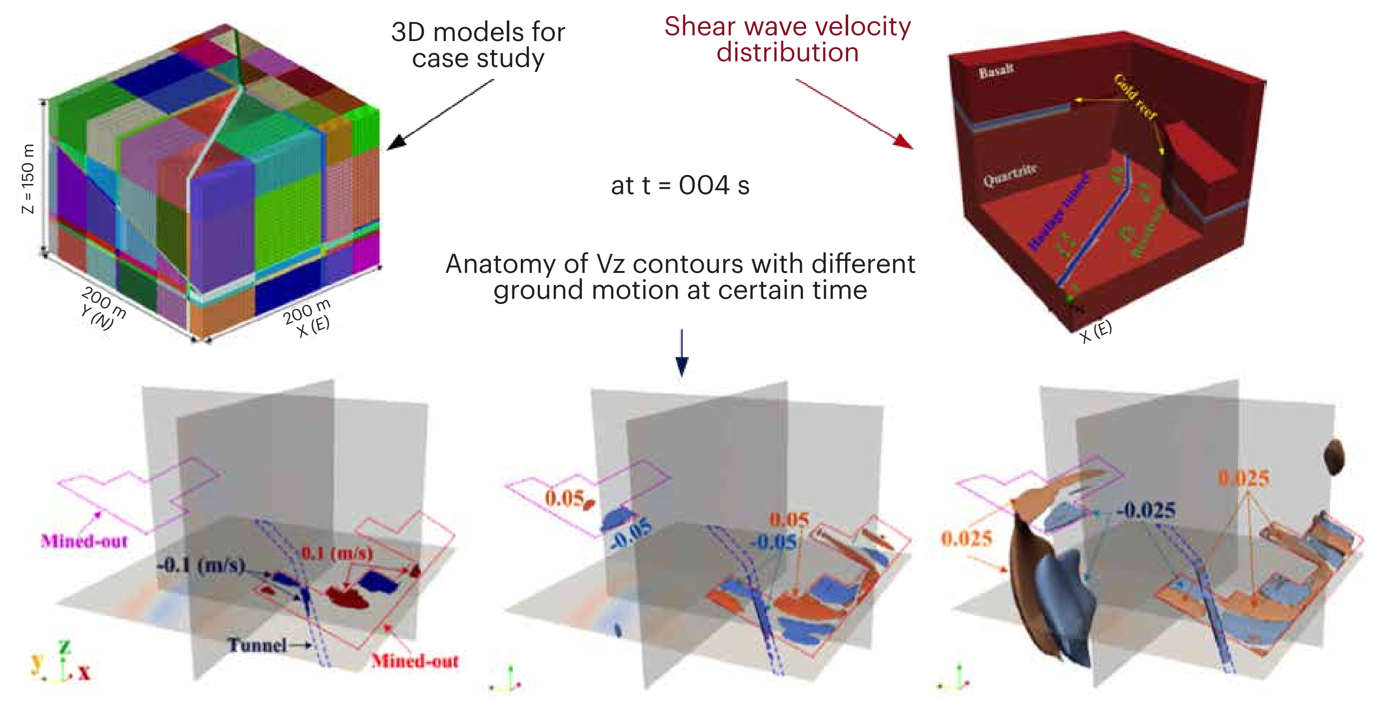

Figure 56: A case study at a gold mine in South Africa using SPECFEM3D

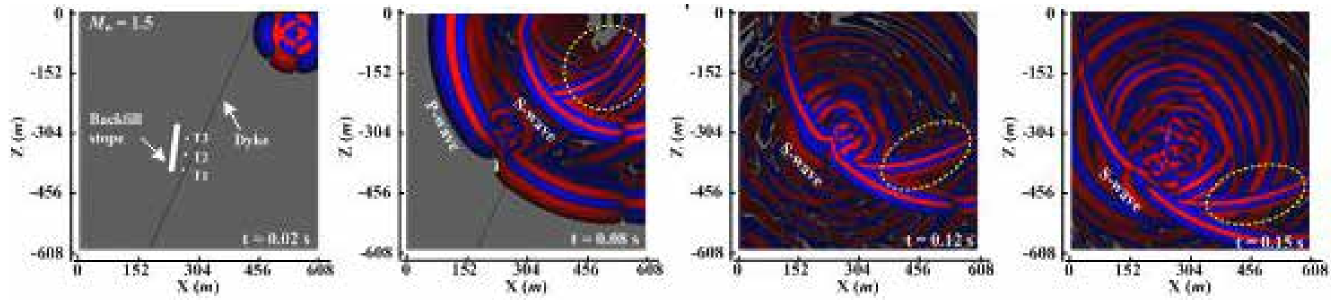

Figure 57: Snapshots of the horizontal velocity components at four propagation times for a model with a tunnel, an open stope, and a dyke. The positive and negative values of the horizontal velocity (right and left movements) are represented by red and blue, respectively

achieved if a non-linear velocity model based on the proposed method is used. This can enhance the usefulness of microseismic monitoring systems in deep underground mines.
- The developed coupled numerical technique can be used to assess PPV around excavations more accurately. This in turn can assist safe and cost-effective ground support design.

Publications:

Wang, X., Cai, M., 2015. Influence of wavelength-to-excavation span ratio on ground motion around deep underground excavations. Tunnelling and Underground Space Technology 49, p.438-453.

Wang, X., Cai, M., 2015. FLAC/SPECFEM2D coupled numerical simulation of wavefields near excavation boundaries in underground mines. Computers and Geoscience (Submitted).

Cai, M and X. Wang (2017). Numerical Modeling of Ground Motion near Underground Excavation Boundaries. Chapter 3 in “Rockburst: Mechanisms, Monitoring, Warning and Mitigation”, Editor Xia-Ting Feng, Elsevier (in print; ~40 pages).

NOTE: for a complete listing of presentations and publications please go to:
https://scholar.google.ca/citations?user=kHg_k5kAAAAJ&hl=enn

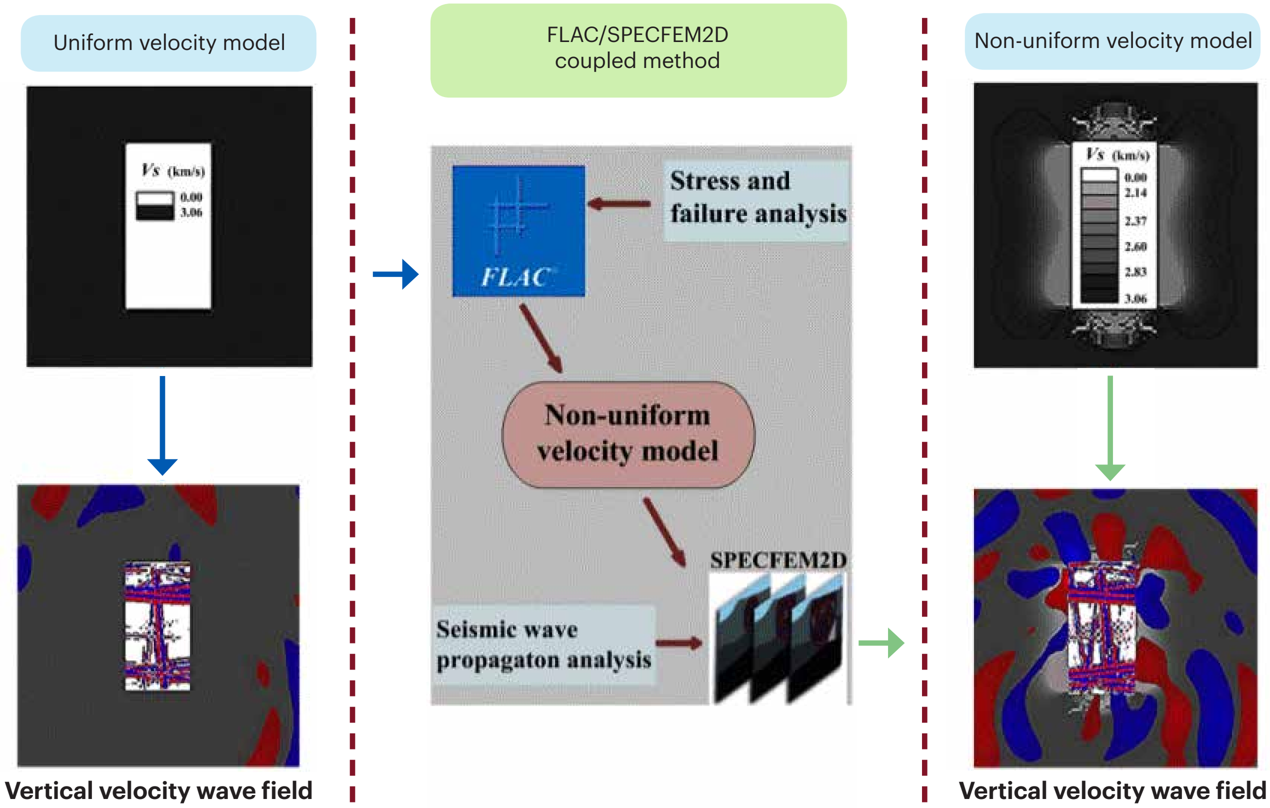

Figure 58: FLAC/SPECFEM2D coupled modeling for uniform and non-uniform velocity models

SUMIT #4

Rockmass Behaviour and Mining-Induced Changes

Rockmass Behaviour and Mining-Induced Changes

Objectives

As indicated in the technical context section of the SUMIT Program, as mines progress to greater depths, the ground becomes less forgiving and the response of the rock mass to mining and the related rock mass behaviour, particularly brittle rock behaviour, must be understood, anticipated and controlled. For this, the in situ and mining-induced states of stress and the rock mass strength must be anticipated and the performance of underground excavations predicted. Once underground access is available, monitoring data can be used to verify the prediction models and to refine the design, e.g., the support selection. Some of the aspects covered by Theme four include:

- Development and implementation of experiments for real-time monitoring of near- and far-field stress/strain change in rock mass (footwall, hanging wall, etc.) together with micro-seismicity rock mass response. Specifically, to understand the response of back filled excavations and to assess and evaluate the role of stiffness changes due to rock mass degradation and failure and many more.
- Investigation of strainburst phenomena and the influence of rock mass and mine stiffness on the strainburst severity.
- Development of the means to determine excavation vulnerability to rockburst damage and to select support for strainbursting ground.
- Assessment of the influence of stiffness changes on the response of highly stressed rock to dynamic loading by investigating rock degradation and damage processes around a borehole under dynamic loading.
- Extension of the concept of mine stiffness to faults and fault networks, for sill pillar design, drift development and extraction and risk reduction strategies.
- Investigation of hydraulic injections to characterize rockmass and stress state in a hard rock mining.

4.1 ROLE OF LARGE SCALE HETEROGENEITIES ON IN-SITU STRESS AND INDUCED STRESS FIELDS

Team:

Laurentian University, MIRARCO – *Drs. Peter K Kaiser, Salina Yong* and *Sean Maloney*

Abstract

At ARMA's 2006 symposium, Maloney et al. (2006) presented a "Reassessment of in situ stresses in the Canadian Shield" to assist in establishing representative in-situ stress conditions appropriate for sub-regional modeling activities. By use of a conceptual heterogeneous rock mass model, it was demonstrated (a) how earth's crust straining may affect the in-situ stress profile, (b) how the in-situ stress magnitudes vary in heterogeneous rock masses, and (c) what the consequences are for excavation stability in such variable stress fields. The first two points are discussed by presenting a reinterpretation of the data set published by Corthésy et al. (1998). For a hypothetical circular shaft with a defined rock mass profile, it is illustrated how large-scale heterogeneities modify the induced stresses at the excavation boundary and thus affect longitudinal variations in the depth of failure. It is concluded that the combination of stress and strength heterogeneities leads to a highly variable excavation behaviour with localization of various failure modes. Most importantly, it is demonstrated that the common assumption of far-field stress boundary conditions may lead to non-conservative model predictions when compared to far-field strain boundary conditions.

Context and Methodology

Natural rock masses are heterogeneous in terms of modulus and strength across all length scales and this heterogeneity modifies the in situ and excavation-induced stress field. In rock engineering, it is common practice to assume that the stress field is uniform and can be described by a single stress tensor. As a consequence, variations in excavation instability would have to be attributed to variations in rock mass strength alone. Experience by the lead author during a recent litigation case, which unfortunately cannot be published, has shown that the stress field was highly modified by rock mass strength and modulus heterogeneities and that the excavation behaviour changed drastically as a result of this variability in strength and stress. Without reference to this particular case, simplified numerical (Voronoi) models, calibrated on published data (Corthésy et al., 1998), were used to illustrate that:

- variations in field measurements can be attributed to rock mass stiffness variations;
- mining-induced stresses along an underground project can be highly varied; and the
- variability in excavation behaviour has to be attributed to variations in both strength and stress.

In particular, it was demonstrated that heterogeneous rock masses, in terms of rock mass modulus, affect the local stress field differently under various far-field boundary conditions such as sedimentary, thermal, tectonic, and mining-induced strain boundary conditions.

Summary of Findings

The principal stress data presented by Corthésy et al. (1998) was reinterpreted within the framework of near surface rock mass yield and heterogeneity within a thermal or tectonic strain field. A Voronoi model with higher rock mass and joint strengths, allowing yield to a depth of 400 m, is adopted to establish the trends for the Abitibi region. The simulated stress profiles presented in Figure 59 are produced by a plastic, heterogeneous Voronoi model with a random rock mass modulus variation and deformed by constant horizontal straining.

In summary, this figure demonstrates that the data trends are well represented by a strained heterogeneous rock mass model. This reinterpretation of data from the Abitibi area confirms that the often-measured stress variability is reality and can be attributed to rock mass heterogeneity; certainly, in thermally or tectonically strained regions of the world.

The implication of such stress variability on excavation stability was explored for a rock mass where brittle failure dominates the depth of failure around excavations. The excavation-induced tangential stresses for a circular vertical shaft and for horizontal tunnels, excavated in the principal stress directions, at different depths are presented in Figure 60. Also, shown is the randomly generated UCS profile that was used to estimate the depths of failure as a function of depth below ground surface. The profiles of the induced depths of failure for the stress and UCS profiles presented in Figure 60 are shown in Figure 61 for a vertical shaft.

Figure 61 highlights that a wide range in excavation damage must be expected due to the combined effect of stress and strength variation. At shallow depth, where UCS was assumed to be constant, the stress variability strongly influences the variability in the extreme depth of failure. The impact of strength variability is shown for depths exceeding 600 m. For this depth range, the variability due to stress is relatively low and the depth of failure gradually increases from about df/a = 0.6 to 0.8; i.e., between three and 4 m for a shaft with a diameter of 10 m. The UCS variability however dominates at this depth and, as shown by one particular realization of a random UCS distribution, the anticipated extreme depth of failure varies between df/a = 0.3 and 1.3; i.e., locally up to 6.5 m for a 10 m wide shaft.

Conclusions

From the findings presented above, it follows that:

- Stress variability can be attributed to rock mass heterogeneities that become dominant in conditions where the earth's crust is thermally or tectonically strained. Strength and stiffness heterogeneity both strongly affect the variability in the in situ and induced stress profiles.
- Commonly assumed linear trends fitted to stress data are only justified for sedimentary depositional environments.
- The gradient of the minor and major horizontal stress profile is typically much steeper at depth than in the vertical stress profile. As a consequence, the major, intermediate and minor principal stresses may switch orientations. In such cases, it is inappropriate to fit trend lines without considering stress tensor rotations.

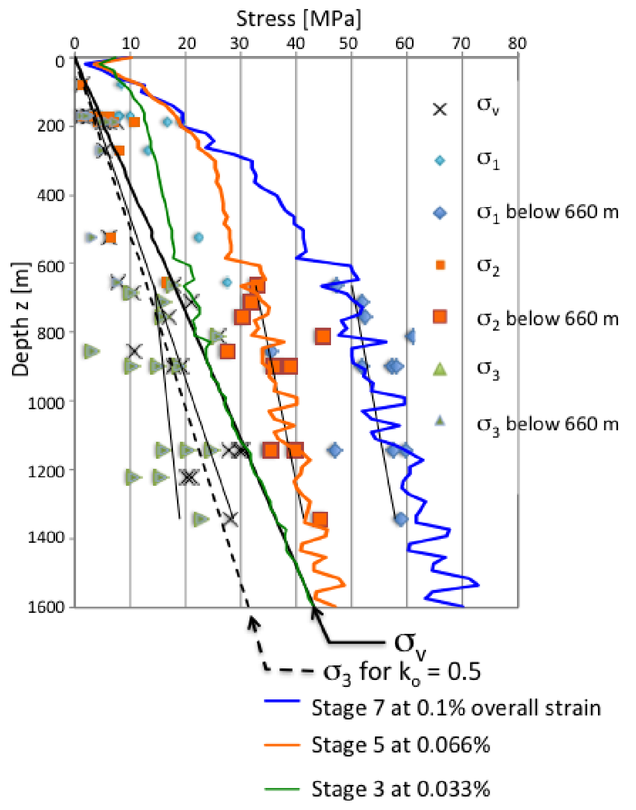

Figure 59: Reinterpretation of data from Corthésy et al. (1998) with a random rock mass modulus variation; three solutions for sH at 0.033 to 0.067 to 0.1% constant overall horizontal strain (green, orange, blue) are shown for heterogeneous, plastic rock block model

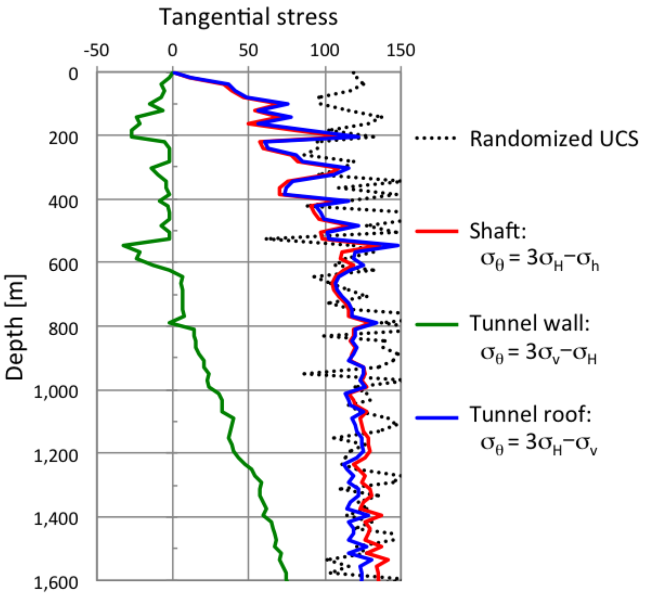

Figure 60: Tangential stress profiles for a vertical shaft (due to sH and sh); for tunnel roofs (due to sH and sv); and tunnel walls due to sH and sv. A randomly generated UCS profile for UCS = 125 MPa with a CoV = 20% is also shown

- At shallow depth, stress variability tends to dominate tunnel or shaft instability by either causing relaxed conditions or by generating rapid changes in the depth of failure profile.
- At depth, the extreme depth of failure still varies due to stress variability and gradually increases with depth. The variability in rock strength however tends to dominate the depth of failure at greater depth.
- Meaningful in-situ stress numerical models can be obtained for rock masses with large-scale rock heterogeneities by adopting horizontal strain rather than stress boundary conditions. As a matter of fact, the common assumption of far-field stress boundary conditions may lead to non-conservative model predictions when compared to far-field strain boundary conditions.

Selected References

Corthésy, R., D.E. Gill, and M.H. Leite 1998. Élaboration d'un modèle de prédiction des contraintes in-situ dans la région de la faille de Cadillac. CIM Bulletin 92(1020): p.54-58.

Maloney S., P.K. Kaiser, A. Vorauer, 2006. A Reassessment of in situ stresses in the Canadian Shield. Golden Rocks, Proceedings of 41st U.S. Symp. on Rock Mechanics (USRMS), Golden, Colorado, 9p, CD.

Selected Publications:

Kaiser PK, SM Maloney, S Yong 2016. Role of large scale heterogeneities on in-situ stress and induced stress fields. Proc. of 50th US Rock Mechanics/Geomechanics Symposium, Houston, USA, Paper 16-571, 9p.

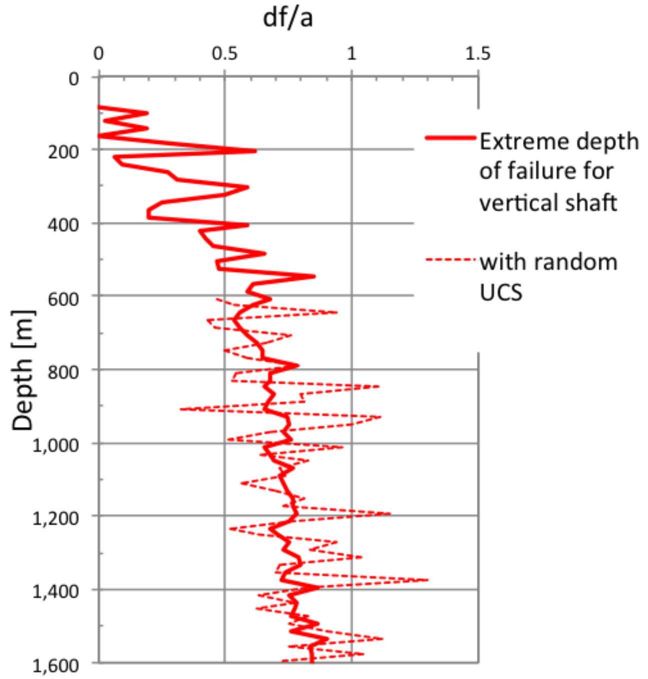

Figure 61: Extreme depth of failure profiles for a shaft with the stress profiles and randomized strength shown in Figure 59

NOTE: for a complete listing of presentations and publications please go to: https://scholar.google.ca/citations?user=BM3D-s4EAAAAJ&hl=en

4.2 Rockmass Stiffness Modification and Post-Yield Behaviour

Team:

Queen's University – *Drs. Mark Diederichs* with *Gabe Walton* and *F. Duran*

Title: Dr. Mark Diederichs
Role: Co-PI
Collaborators on team: Vale, Codelco, Antofogasto Minerias, Sudbury Integrated Nickel Operations – A Glencore Company

Abstract

This work builds on earlier reported work in this volume and improves our understanding and ability to simulate both rock damage and dilation feedback to strength during complex stress paths and to accurately predict and model near excavation deformations, a critical aspect of support design for deep mining.

Context

This subproject had two themes – stress path dependent degradation (damage) and damage induced dilation. In fact, the two projects addressed the evolving strength, stiffness and post-damage and post-yield behaviour (prior to actual rock failure). In the work of Duran (et al), a complex synthetic rockmass model composed of a bonded disc model for the rock matrix and discrete structural discontinuities was used (Figure 62). The study included validation of this approach for advanced modelling but also investigated the role of stress path on pre-peak damage and strength loss (Figure 63).

Summary of Findings

This work resulted in the development of a new continuum dilation model that could be used in more simplified conventional analysis (a state-of-art addition to state-of-practice modelling). This dilation model (Figure 64) was developed and verified through lab testing as well as numerous field analysis.

Further verification of the model and investigation of the implications for analysis and design related to both pillar skin (supportable rock) and pillar core (supporting rock) was carried out in an experiment at Creighton Mine (Figure 65, 66 and 67). Here the model was validated with Extensometer and LiDAR data (from an earlier piece of work reported in this volume) and then used

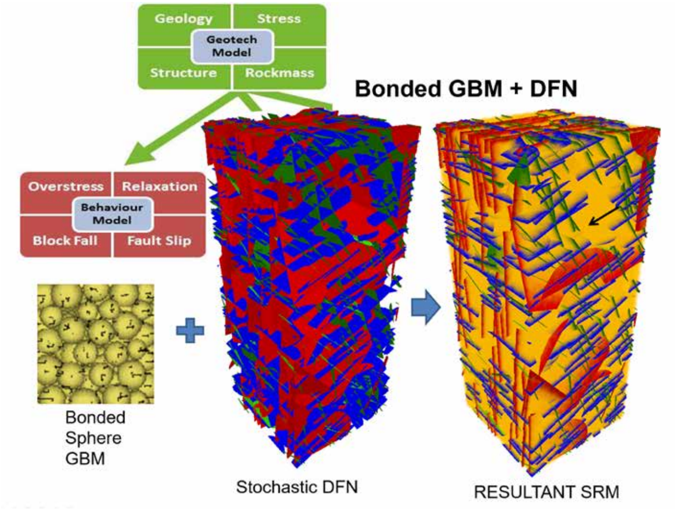

Figure 62: Synthetic Rockmass Models for damage analysis of rockmasses (Duran et al)

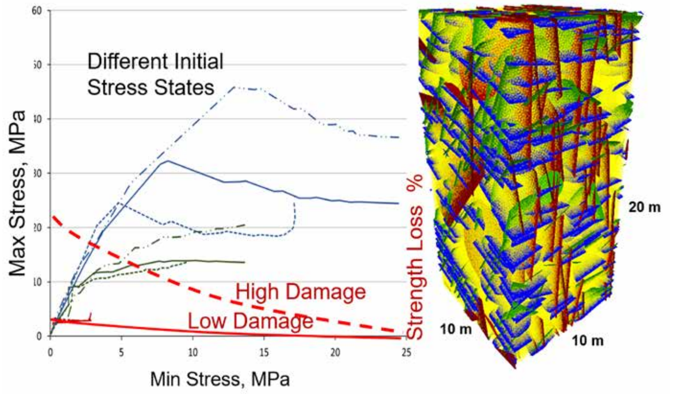

Figure 63: Dependence of damage evolution on stress path from SRM (Duran et al)

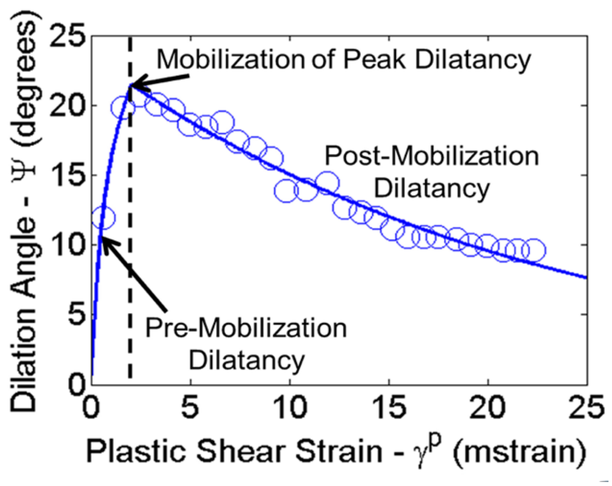

Figure 64: Dilation model for continuum analysis or rock (Walton et al)

to understand the true nature of damage evolution and self-stabilization of rock pillars. This understanding represents a major step forward.

Practical Implications

The contributions of this project and the integrated work at Queen's Geological Engineering have led to advances in our understanding of rock behaviour and to create more realistic models for analysis, and in our capacity to assess the inputs and interpret the outputs of such models for design and Geo-risk reduction.

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Walton, G. 2014. Improving continuum models for excavations in rockmasses under high stress through an enhanced understanding of post-yield dilatancy. PhD Thesis, Queen's University.

Walton, G. and Diederichs, MS. 2012. Comparison of practical modelling methodologies for considering strain weakening and dilation as part of geomechanical analysis. . In Proceedings of RockEng12: 21st Canadian Rock Mechanics Symp. Edmonton. May. Paper No.: 7113. 8p.

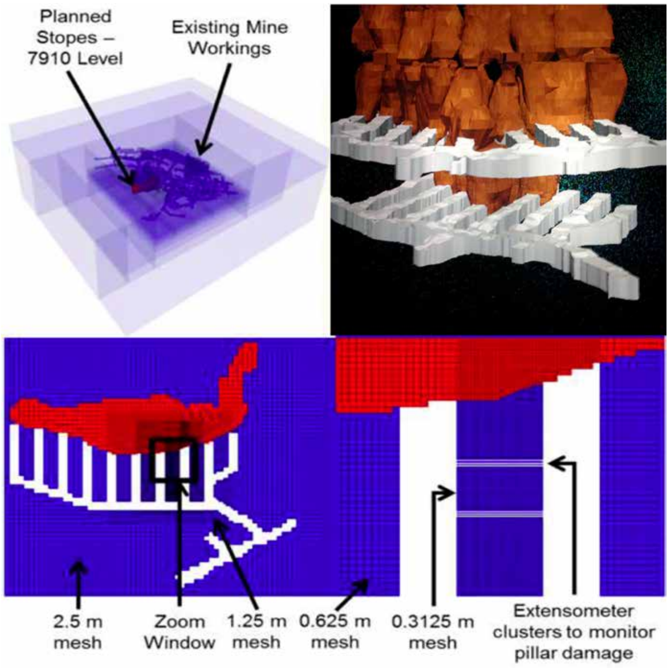

Figure 65: 3D model setup of Creighton deep level analysis (Walton et al)

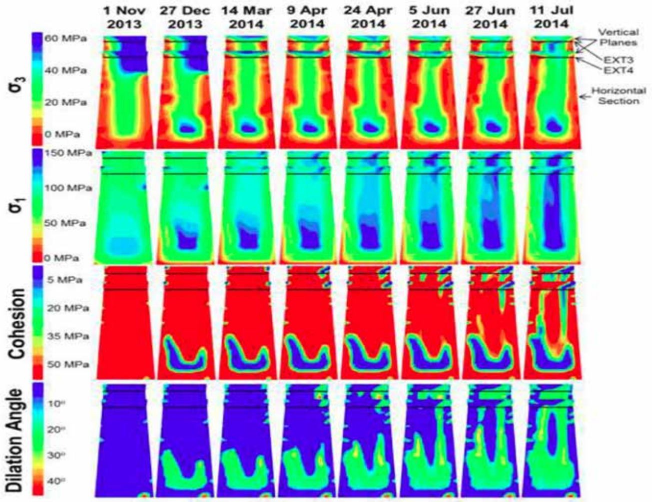

Figure 66: Evolution of pillar stress and strength parameters during mine-by (Walton et al)

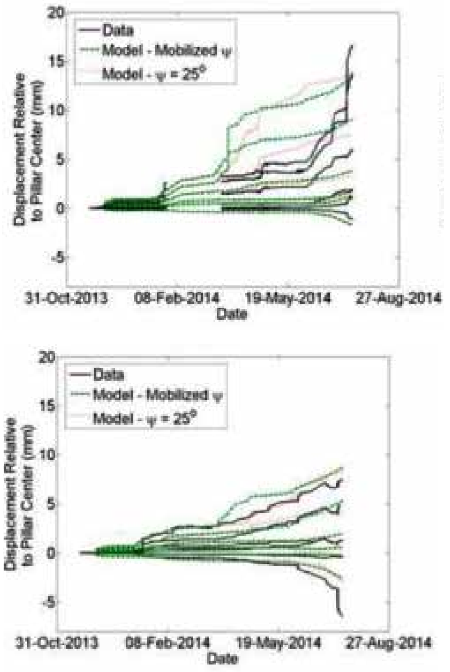

Figure 67: Comparison of model results with measured pillar deformations (Walton et al)

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4.3 Shear Rupture Evolution Causing Rockbursting in Mines

Team:
University of Toronto and Laurentian University, MIRARCO – *Drs. Rob Bewick, Peter K. Kaiser* and *Will Bawden*

Abstract

A fault-slip rockburst is damage predominantly caused by the energy radiating from a dynamically slipping, pre-existing fault, fault zone or from a newly generated shear rupture. This investigation focussed on the creation of shear ruptures during mining. A grain based Distinct Element Method and its embedded Grain Based Method was used to simulate the fracturing processes leading to shear rupture zone creation in a calibrated massive (non-jointed) brittle rock deformed in direct shear under constant stress and constant normal stiffness boundary conditions. Under the latter boundary conditions, shear rupture zone creation relative to the shear stress versus applied horizontal displacement (load–displacement) curve occurs pre-peak, before the maximum peak shear strength is reached. This is found to be the result of a normal stress feedback process caused by the imposed shear displacement which couples increases in normal stress, due to rupture zone dilation, with shear stress, producing a complex normal-shear stress-path that reaches and then follows the rock's yield (strength) envelope. Once the maximum peak strength is reached (after a series of shear stress oscillations) the largest stress drops occur as the ultimate or residual shear strength is approached. The simulation results provide insight into the fracturing process during rupture zone creation and improve the understanding of the shear stress versus applied horizontal displacement response, as well as the stick-slip behaviour of shear rupture zones that are being created under constant normal stiffness boundary conditions.

The studies demonstrate that differences in seismic patterns can be attributed to prevailing mining-induced boundary conditions. The practical implications are that precursory seismic events differ depending on these boundary conditions.

Context

A critically stressed fault, with shear stresses reaching its shear strength, can slip, particularly when the degree of freedom is changed as it is intersected by a mine opening or by the yield zone surrounding a mining area. The strength of a fault is a function of the normal stress, the coefficient of friction of the fault surface, its waviness or dilation characteristics, and in the strength of rock bridges, called asperities in the earthquake literature. A fault may therefore slip, for example, when the shear strength is reduced due to a drop in clamping stress or the failure of asperities (rock bridges) occurs.

Damage can also be caused by shear rupture through massive or moderately jointed rock masses as illustrated by the image captured by Ortlepp (1997) after mining through an area that previously experienced fault-slip events (Figure 68).

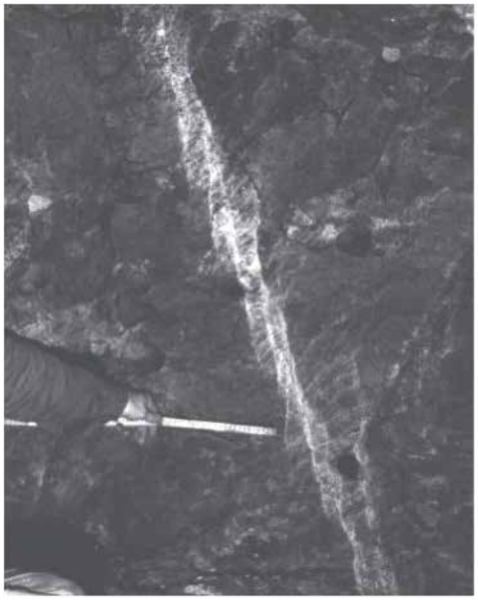

Figure 68: Ruler pointing at shear rupture, known as Rupture No. 18 (Ortlepp, 1997)

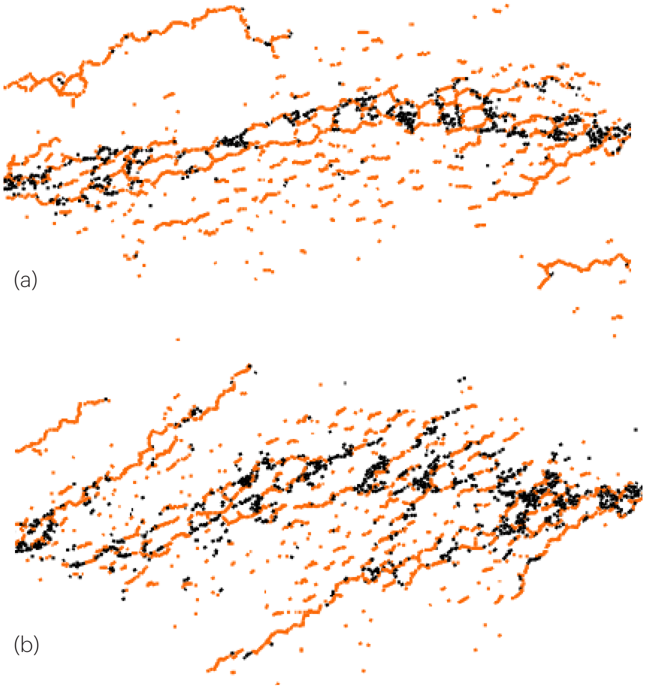

Figure 69: Shear rupture created in a soft, dilative environment under relatively low and high normal stress (Bewick et al., 2014)

Such shear rupture rockbursts have been observed particularly in South African mines (Ortlepp, 1997; Ortlepp, 2000) with Richter magnitudes exceeding mL = 3.5. Ortlepp (1997) advocated shear rupture as one of the most important source mechanisms for major rockbursts.

Summary of Findings

The research by Bewick et al. (2013) shows that the characteristics of a shear rupture zone are not only a function of the rock or rock mass properties but also of the prevailing boundary or confining conditions under which a rupture zone is created. In soft environments, shear rupture is facilitated by the ease of dilation. An example of shear rupture creation from a numerical model is shown in Figure 69 for low and high normal or confining stress conditions.

Shear ruptures are localized at low normal stress and become increasingly complex with en echelon fractures making up a wider shear zone when created at higher normal or confining stress as shown above. It should be noted that the ultimate rupture zone is not planar, is embedded in a relatively wide rock mass damage zone, and may be composed of associated en echelon shears which are themselves faults (if so, seismic event focal plane solutions will not be aligned with the direction of the rupture zone during early stages of creation. Only once a shear rupture is well developed, having undergone significant displacement would seismic event focal plane solutions be aligned with the slipping direction.

As a general guide, shear ruptures are created post-peak at low normal stress, display more post-peak brittleness, and thus could be more prone to violent failure (Bewick et al. 2016). But due to the brittleness, the shear rupture would be associated with only a single large event. At high normal stresses, repeated seismic events may be expected when the rupture zone is developing due to the failure process generating multiple en echelon shears to create the shear rupture zone also post-peak strength.

In stiff environments where dilation is constrained, coupling between the normal and shear stresses (stress-path) under constant normal stiffness boundary conditions leads to a different rupture development process. The shear rupture creation process is displacement rather than stress controlled and occurs pre-peak strength. Due to the displacement controlled process, ongoing mining-induced deformation is needed to fully form shear ruptures in stiff environments. As illustrated by Figure 70, the eventual shear zone is localized and also associated with en echelon fractures.

Most importantly, large stress drops (seismic events) can be expected long before the shear rupture has formed, i.e., even before the peak strength of the rupture zone has been reached. Repeated larger stress drops or shear stress oscillations are encountered due to cohesion loss during the rupture zone formation. Hence, repeated seismic activity is indicative of the formation of a shear rupture zone in a stiff environment.

Fault-slip and shear rupture rockbursts typically occur in deep mines when the extraction ratio is high and large closures are allowed to persist over large mining volumes. Associated fault-slip events may release large amounts of seismic energy, coming from the instantaneous partial relaxation of the elastic strain energy stored in a large volume of highly stressed rock surrounding the slip or rupture area, and radiate seismic energy in the form of compressive (p) and shear (s) waves. These ground vibrations or ground motions may be triggering strainbursts (see Section 5.1, p. 61) or pillar bursts, may cause dynamically loaded strainbursts or shakedowns, or eject insufficiently supported rock by energy momentum transfer to broken rock.

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https://www.researchgate.net/profile/William_Bawden/publications for Bawden

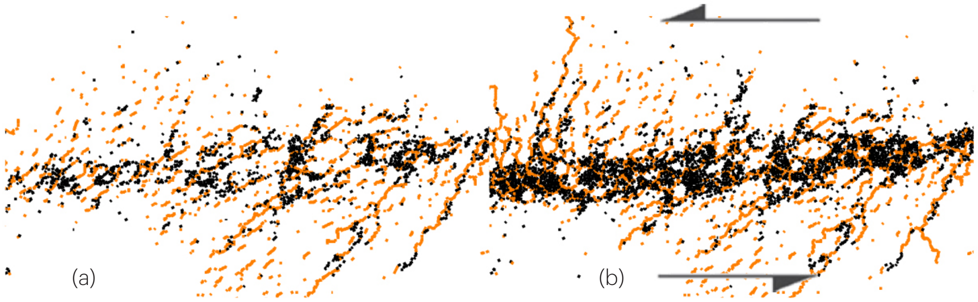

Figure 70: Shear rupture created in a stiff, non-dilative and highly confined environment (Bewick et al., 2014)

4.4 Hydraulic Injection for Rock Mass and Stress Characterization

Team:
University of Waterloo – *Dr. Maurice B. Dusseault* and *Atena Pirayehgar*

Title: Dr. Maurice Dusseault
Role: Co-PI
Collaborators on team: Vale, NSERC and SMARTEC

Project Goals

This project aims to better understand the deformation mechanics of hydraulically induced fractures in the presence of natural fractures.

Abstract

Hydraulic fracturing of naturally fractured rock masses is used in mining to precondition rock before controlled blasting for caving design. Studies of this using DEM (Discrete Element Models – UDEC) show the effects in the presence of a room, delineating the shape and extent of the zone most greatly affected by the HF activity. The DEM modeling has also shown how stick-slip behaviour of joints in the broader area around a change in pressure (injection point) can be studied so that one may better understand the impact of natural rock structure and stress fields on induced microseismicity patterns. A key aspect of understanding the processes is to quantify effects related to rate or viscosity so that better choices of HF methodologies for rock mass conditioning can be made.

Context

Hydraulic fracturing as a treatment method to aid mining has been suggested for several decades in the coal and metal mining industry. Different applications have been suggested as being potential contributions to safety and economic benefits. Recently, hydraulic fracturing has been applied to precondition rock or to induce caving in hard rock underground mines.

Fractures during mining are conventionally induced by blasting to help the broken rock mass flow from conditioned stopes under gravity forces. More and more, massive caving processes are used to extract entire ore bodies, involving large-scale rock mass fragmentation through massive stress redistribution, and leading to induced caving and flow for the entire orebody, while trying to reduce the amount of blasting needed and also reduce risks of uncontrolled bursts and inadequate fragmentation. Preconditioning of the rock mass to aid this process via hydraulic fracturing involves pre-weakening of orebodies with low friability that are not caved easily. Hydraulic fracturing also provides the possibility of stress relief and redistribution, reduction in the stiffness of rock masses, and other associated effects that can increase rock mass caveability to allow it to cave continuously in a controlled stable manner.

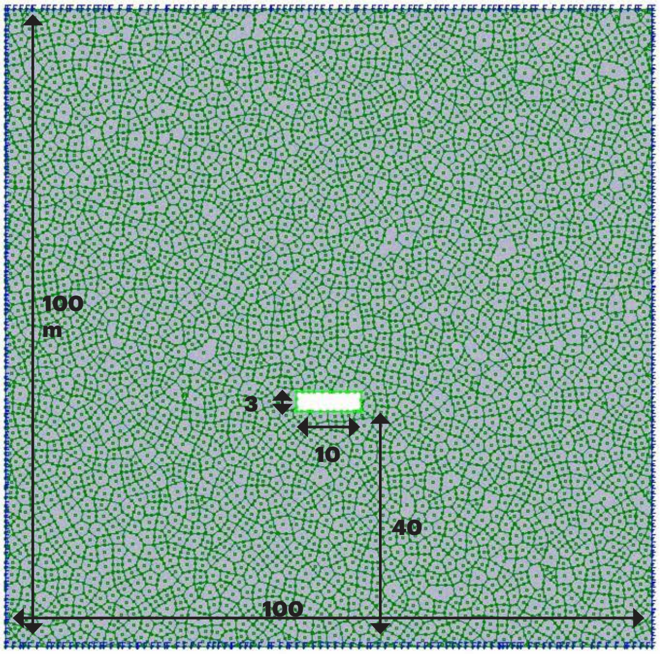

Figure 71: Simulated undercut in the block-caving model with extended opening joints

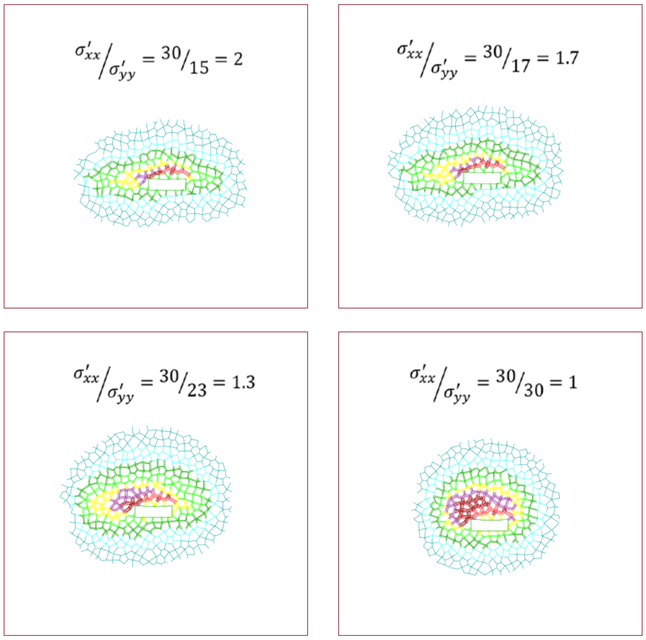

Figure 72: Pore pressure distribution around the injection point #1 under different stress states

Methodology

It is of interest to numerically simulate hydraulic fracturing as an aid to underground mining. UDECTM software was used to simulate the behaviour of a two-dimensional jointed rock mass with a defined undercut under biaxial in-situ stresses.

UDEC is used to analyze fluid injection under constant pressure (steady-state flow) through the fractures in an rock mass with impermeable blocks, which represents a naturally fractured hard rock or stiff shale. A coupled hydro-mechanical analysis is applied via the UDEC software to model both rock and fracture behaviour in the HF/NFR system.

In the current study, a Voronoi tessellated continuum has been generated to evaluate the effect of the stress ratio on flow into the joints by changing differential principal compressive stresses. Given the difference in in-situ stresses, pore pressure distribution is monitored and the distribution of slip and opening of fractures at different stress field anisotropy is investigated during the pressurized hydraulic injection.

Discrete element analysis can be applied to study processes in naturally fractured rock masses. It is possible to simulate a block

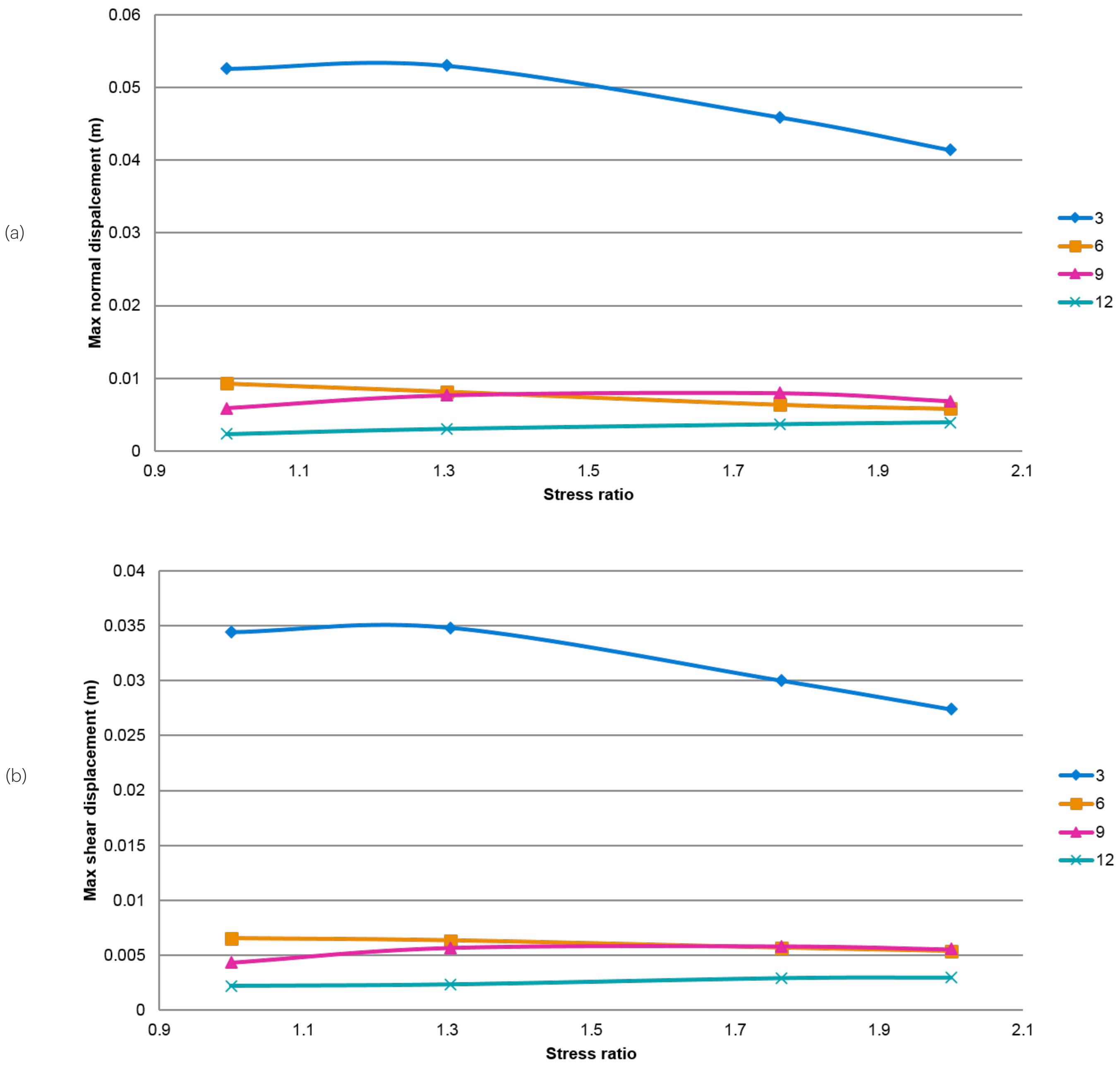

Figure 73: Maximum a) normal and b) shear displacements vs stress ratio for all injection points

with a defined undercut in a jointed rock to examine the response of the overlying rock in an attempt to emulate a continuous caving process. The analysis required the development of a suitable representation of the rock mass as a continuum cut with joint surfaces in appropriate directions, and this preliminary model is subjected to boundary loads and allowed to reach a condition of static equilibrium under the in-situ stresses. In a real situation, an undercut is created below the orebody and blasting carried out to initiate caving, a process that might be simulated by the creation of additional fractures in the 2D model, and the flow of these rock blocks can be tracked. Here, we examine hydraulic fracturing processes.

Summary of Findings

Figure 71 shows a 2D plane-strain section generated in UDECTM to represent a fractured rock mass 100 m × 100 m in size, subjected to a biaxial in-situ stress state. The medium is initially dry and then water is injected at a constant flow rate of 0.4 L/s at a point in the model.

The effect of stress ratio on flow into the joints has been studied as a function of injection distance from the top of the undercut. Given the difference in situ stresses, joints re-open under the induced pressures dominantly parallel with the maximum principal stress, a consequence of work minimization However, in the isotropic condition, no explicitly favored path exists for fractures to propagate, so the fluid pressure has a similar effect all around the injection point, excluding the local fabric effects. Figure 72 shows the pore pressure distribution around the injection point for different stress ratios.

Further, it seems that there is an effective injection distance after which rockmass response remains approximately constant. Applying hydraulic fracturing further from the effective distance would precondition the hard rock but it takes longer to reach the caved block or a higher injection pressure is required. This is an important scale effect that must be considered in the preconditioning design process.

Figure 73 indicates maximum normal and shear displacements after injection at different points in the 2D fractured continuum subjected to an isotropic and a differential stress (the horizontal stress is kept constant for different simulations). Blue lines show the largest magnitudes of normal and shear displacements; these values belong to the injection at the closest distance from the cave back, and they decrease under stronger stress differences. The proximity of the undercut back to the injection points and an increase in the applied compressive stress lead to larger displacements. Far from the undercut back, displacements have smaller magnitudes with a slight increasing trend with larger deviated stress, which has been discussed before (Pirayehgar & Dusseault, 2014). Stresses therefore control the deformation, and the effect depends strongly on the location of the injection point up to some distance.

Practical Implications

Numerical simulation suggests that a clear understanding of both the stress ratio and of where to place injection holes is needed to ensure optimized fracture creation for preconditioning results.

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for a complete listing of the principal authors’ publications please go to:
https://www.researchgate.net/profile/Maurice_Dusseault

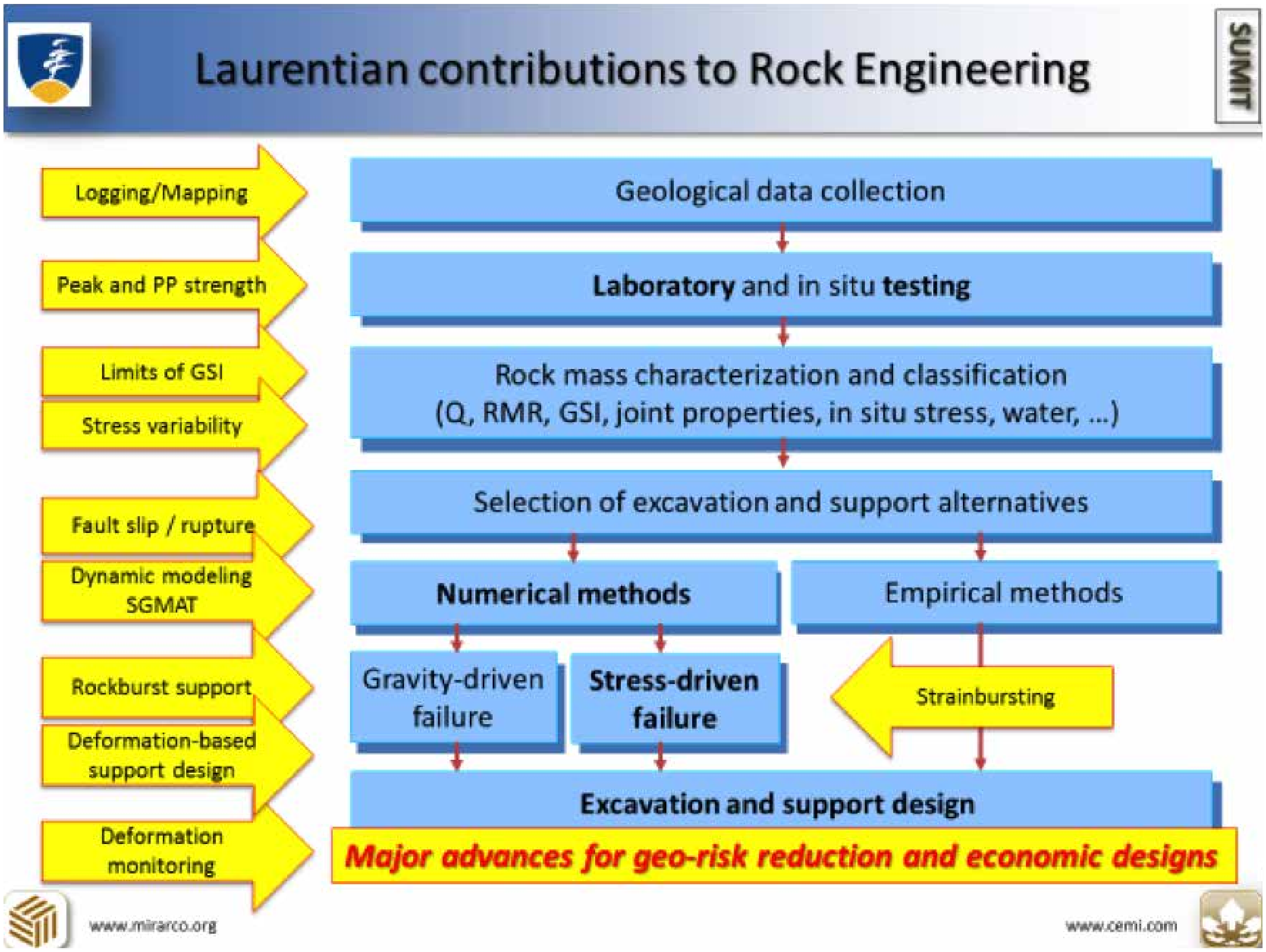
SUMIT #5

Enhanced Mine Development – Underground Mine Construction and Design

Enhanced Mine Development – Underground Mine Construction and Design

Objectives

The engineering design process involves many steps, as indicated in the following flow chart which highlights advancements made by the Laurentian University team to this process. The research under this theme aimed at making contributions, in combination with the research described under Theme 2, to reduce the geo-risk and to arrive at cost-effective designs.



Laurentian contributions to Rock Engineering

SUMIT

Logging/Mapping

Peak and PP strength

Limits of GSI

Stress variability

Fault slip / rupture

Dynamic modeling SGMAT

Rockburst support

Deformation-based support design

Deformation monitoring

Geological data collection

Laboratory and in situ **testing**

Rock mass characterization and classification
(Q, RMR, GSI, joint properties, in situ stress, water, ...)

Selection of excavation and support alternatives

Numerical methods

Empirical methods

Gravity-driven failure

Stress-driven failure

Strainbursting

Excavation and support design

Major advances for geo-risk reduction and economic designs

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Of the wide spectrum of possible research projects, this part of the SUMIT Program focussed on dynamic processes (e.g., strainbursting), on monitoring technologies (e.g., fibre optics), on stress management (e.g., hydraulic pre-conditioning) and on the selection of effective support systems.

For these purposes, researchers would aim to develop innovative methods and procedures to understand and quantify strainburst damage processes and to select effective dynamic ground support systems for more effective mine design with consideration of constructability matters.

As well, numerical modelling and synthetic ground motion tools would need to be developed and used to better assess dynamic excavation damage processes and assist in support design. With respect to monitoring technology, two university teams aimed to focus on the development and adaptation of distributed fibre-optics based strain monitoring and tiltmeter arrays applications to measure rock mass strain (deformation), and the development of Fibre Bragg Grating (FBG) based broadband strain monitoring arrays with inclination sensors for deep mining application. Investigations on hydraulic preconditioning as means of rock mass damage to mitigate energy release from highly stressed ground would be initiated and assessed by numerical modelling. Finally, both modelling and field instrumentation projects would be undertaken to assess both pillar loading processes and pillar failure processes in brittle rocks.

5.1 Strainburst Identification and Control for Underground Mine Construction

Team:
Laurentian University, MIRARCO – *Dr. Peter K Kaiser*

Abstract

Two aspects of ground control in burst prone ground were addressed by this project: 1) the critically important issue of assessing the vulnerability of underground excavations to damage by dynamic disturbances, and 2) excavation damage caused by strainburst.

The first part aimed at overcoming one of the current fundamental flaws in ground control for rockburst damage mitigation, i.e., a lack of systematic methods to assess the vulnerability of excavations to dynamic disturbances. Most support selection approaches, whether risk-based or following systematic engineering design methods, do not properly take into account the vulnerability status of an excavation before it is affected by a dynamic disturbance. Kaiser (2017) will provide guidance on how to assess the vulnerability or safety margin for three primary damage processes: seismically induced falls of ground or shakedowns; stress-driven strainbursting; and energy transfer from remote seismic events. Emphasis is placed on the second, strainbursting, because common ground motion-centric approaches are particularly flawed when dealing with strainbursting ground. The result of this part of the work will be published in Kaiser (2017).

The second part aimed at advancing the state-of-the-art in anticipating strainbursts, i.e., the strainburst potential (SBS), and the damage severity resulting from strainbursts, i.e., the strainburst severity (SBS). The result of this part will be published in Cai and Kaiser (2017) and presented during short courses provided to industry starting in May 2017 in Sudbury.

Context

Part 1: Heal et al. (2006) established that the response of an excavation to a dynamic disturbance and the severity of damage depends on the vulnerability of an excavation to failure. As explained in detail by Kaiser (2017), this approach is not suitable without modification to fully assess the vulnerability of an excavation to damage by all types of dynamic disturbances. Their work is however, highly relevant as it recognizes the importance of the excavation vulnerability, i.e., that identical dynamic events will cause a drastically different damage type and extent depending on an excavation's vulnerability to failure. These deficiencies have been overcome by the research at Laurentian University and are reflected in Kaiser (2017).

Part 2: The vulnerability to instability caused by self-initiated or triggered strainbursts was not sufficiently recognized in the past. Whereas it was understood that the burst potential depends on

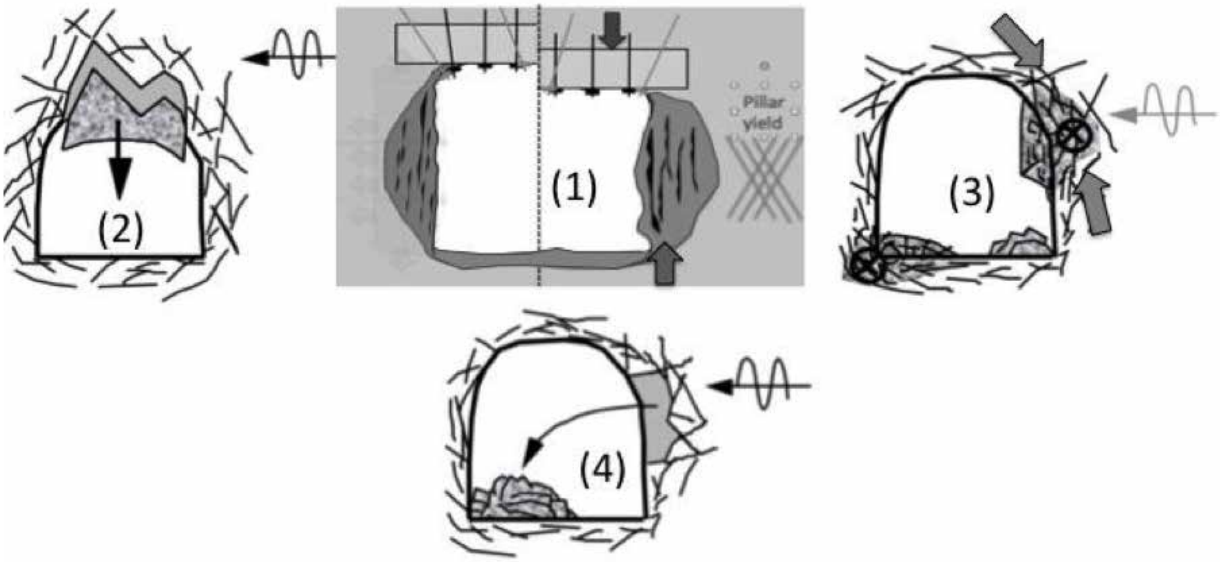

Figure 74: (1) Illustration of mining-induced tangential straining of the drift wall due to floor heave or roof sag (roof loading or pillar yield) and predominant dynamic failure modes near excavations: (2) shakedown or FoG due to acceleration forces from remote seismic event; (3) static stress fracturing or strainbursting due to tangential straining; and (4) rock ejection by momentum transfer from remote seismic source or high bulking deformation rate during strainburst

stress to strength ratio, no systematic approaches were available to assess the damage severity of strainbursts, i.e., how much energy is being released during a strainburst and how this energy is consumed during the failure process, i.e., how much energy is consumed by the rock mass and how much by the support components.

Summary of Findings

The means to quantitatively assess the excavation vulnerability for the failure modes described in Figure 74 were developed and presented in Kaiser (2016). It is now possible to systematically assess the safety margin before an excavation is affected by a dynamic disturbance from a seismic event. This allows one to identify locations in a mine that are more likely to fail and thus require enhanced ground support.

A strainburst is a sudden and violent failure of rock near an excavation boundary as illustrated by Figure 75. It is necessary to distinguish between different strainburst types:

A self-initiated strainburst is a rockburst caused by a gradual weakening of the rock mass such that the local stress after some time exceeds the rock mass strength in a relatively soft loading/mining system, i.e., the local mine stiffness has to be softer than the post-peak behaviour of the failing rock mass. Damage is primarily related to the stress level and the volume of bursting ground.

A mining-induced strainburst is a rockburst caused by mining-induced displacements or strain that change the local stress near an excavation such that the stress (temporarily) exceeds the rock's strength. Damage is also related to the stress level and the volume of bursting ground.

A seismically triggered strainburst is a self-initiated or a mining-induced strainburst that is triggered by a remote seismic event. In this case, the remote event is the primary seismic event and the seismic event co-located with the strainburst damage is a secondary event. Damage is not related to the intensity of the remote event, it only serves as the trigger of a strainburst.

A dynamically loaded strainburst is a strainburst that is augmented by the impact of energy radiated from a primary source in two possible forms:

- the radiating energy causes a dynamic stress pulse that may deepen the depth of failure, thus release more stored energy and, through rock mass bulking, add additional strain or displacement to the rock and support; or
- the radiated energy may transfer some of its radiated energy to kinetic energy and eject part of marginally stable rock.

A strainburst may also occur in a well-supported rock mass, i.e., behind mesh or shotcrete, in the reinforced rock or behind the supported ground. This is called a restrained self-initiated, mining-induced, triggered or dynamically loaded strainburst.

Practical implications

The outcomes from this research are most relevant from a work place safety perspective but also provide the means to better select support in burstprone, particularly in strainburst-prone, ground.

An effective support system must create a deformable reinforced rock wall that can provide and maintain a substantial tangential resistance capacity.

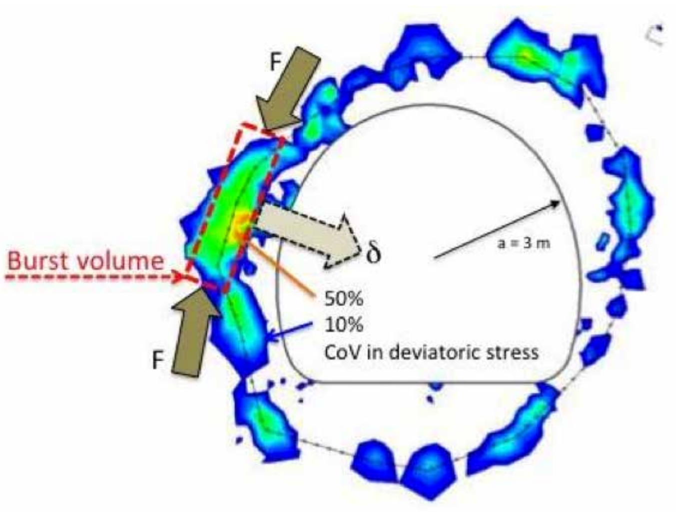

Figure 75: Forces acting on a volume of burst-prone rock and direction of resulting bulking deformation

If a support system is effective in balancing energy releases, it must minimize the bulking in the strainbursting rock mass volume to minimize displacements and strains imposed on the rock reinforcement and the retaining system.

Only if a support system is ineffective and cannot provide sufficient tangential resistance, will the energy component become dominant and eventually excessive. The wall displacements will increase exponentially and may deform at a high rate causing rock ejection.

It is for this reason that deformation-based support design principles must be adopted for the selection of burst-resistant support systems (see section 5.4). Strainburst damage can be prevented or most effectively controlled by a support system consisting of robust retention elements with relatively stiff rock mass reinforcements (that minimize bulking) in combination with yielding bolts that satisfy displacement demand criteria. In other words, the support must be selected to create a deformable, reinforced rock mass in the immediate vicinity of the excavation.

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NOTE: for a complete listing of the author's publications please go to: https://scholar.google.ca/citations?user=BM3Ds4EAAAAJ&hl=en

5.2 Development of a Broadband Sensor Network for Deformation Measurements

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Team:

University of Toronto (see 3.1)

Project Goals

This project aims to identify the state of the art in Bragg grating technology, and devise a reliable means to deploy the technology for grouted borehole dynamic and quasi-static strain measurement underground.

Abstract

Borehole installed-Fibre Bragg Grating sensors to enable stress and deformational measurement changes at the scale of hundreds of metres is investigated as is a reliable method of coupling the sensors to the borehole walls to ensure minimal measurement error resulting from transverse strain. Based upon the results of detailed numerical and analytical evaluations of the mechanical-optical sensor performance, an innovative resin-filled steel casing was designed to properly couple the sensor within a grouted borehole. In this way, the error resulting from transverse strains was minimized.

Context

Mine design has traditionally focused on local rock behaviour and support (adits, shafts, stopes, shotcrete and mesh, etc.). However, larger and deeper mining activity is associated with stress changes at scales far greater than individual stopes or other underground openings. These activities result in stress changes and deformations on the scales of hundreds of metres. A monitoring network is thus needed to effectively detect mining associated variations in the rockmass.

Current mine monitoring techniques, principally microseismic activity records, are limited in their ability to investigate these problems. First, they account for only a tiny fraction of the total energy released by rock deformation and failure (Das and Zoback, 2011). Second and more fundamentally, some destructive failures may occur without microseismic precursors. It is thus desirable to develop a broadband monitoring technique that can measure both the dynamic and the static rockmass deformation. In many

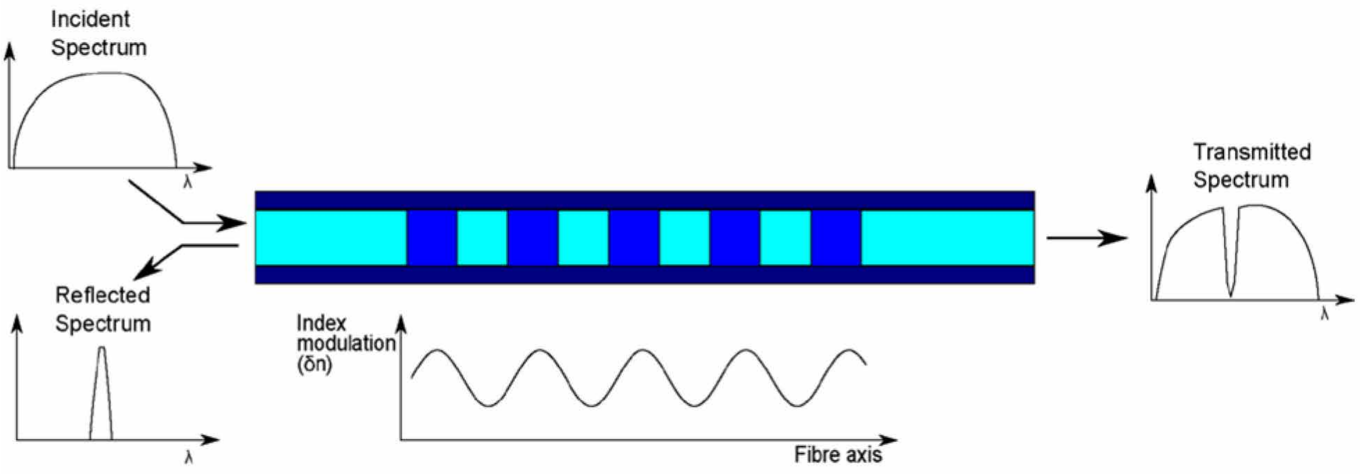

Figure 76: Cartoon of incident spectrum and regular gridding will cause characteristic transmission and reflection spectra

deep mining applications, the evolution of strain in the near and far field is a complex process. The effects of excavation on nearby fault behaviour (for instance induced and triggered seismicity) are driven by mechanisms which have not been fully identified. The study of such phenomena would be improved by accurate and precise measurements made in high temporal and spatial resolution. Fibre optic technology, and fibre Bragg gratings in particular, provide an interesting and potentially powerful means of highly distributed and affordable strain sensing for underground applications.

Methodology

Bragg gratings are intrinsic periodic modulations of the refractive index of a fibre optic core (Figure 76).

Typically ranging from 0.5 to 2 cm in length, the grating has the unique property of reflecting a narrow bandwidth of light when illuminated by a broadband light source. Small changes in strain and temperature alter the periodicity of the grating, thus changing the wavelength of the reflected light. The change, if detected, indicates a change in the measured band of interest.

Since each grating can be written in its own dedicated wavelength band, as many as 20-30 Bragg gratings can be written on a single length of fibre optic cable, thus constituting a quasi-distributed strain and temperature sensing instrument (Figure 77). The FBG strain meter project examined several competing technologies in fibre optic technology which would best accomplish the above stated objectives. It was determined that charge couple device (CCD) spectrography provided the best method for long term strain acquisition, and an ability to detect transient strains with high fidelity and with adequate guards against signal aliasing.

The researchers then undertook a comprehensive numerical and experimental investigation of the sensor performance in grouted boreholes. As fibre Bragg gratings, like most strain sensors, are sensitive to transverse strains, a reliable method of coupling the sensor to the borehole was investigated while attempting to minimise the error induced because of transverse strains, which can cause erroneous measurements.

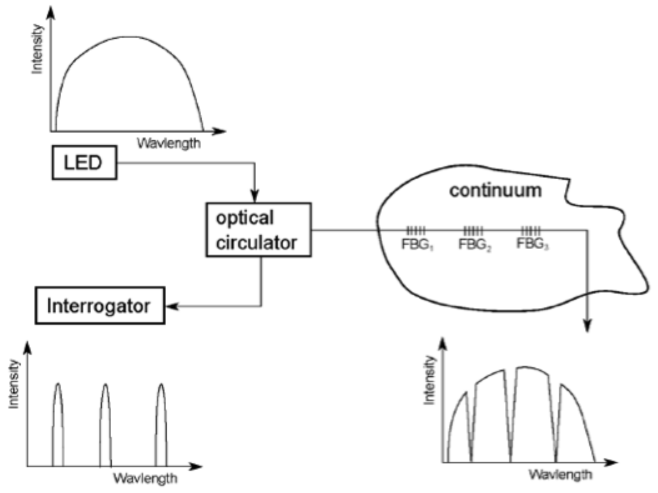

Figure 77: New Bragg-gridding strain meter design for multiple gridding segments (FBG) and multiple spectral peaks and notches

After a detailed numerical and analytical evaluation of the mechanical-optical sensor performance, an innovative resin filled steel casing was designed to couple the sensor to a grouted borehole, while nearly eliminating the deleterious effects of transverse strains on sensor performance. Finally, an experimental investigation was conducted in which the designed casings were used to couple the strain sensor to a rock and grout samples to determine the observed response of the sensor.

Summary of Findings

Results show that it is feasible to detect quasi-static strains within a resolution of one microstrain (με), to an accuracy of 2% using Bragg gratings over a linear range of 2000 microstrain in compression. For dynamic measurements with commercially available CCD technology, the Nyquist frequency is currently set at approximately 2500 Hz. Although the Nyquist frequency is on the lower end of what is typically desired for geophysical data acquisition, CCD technology is an active field of research (for many applications apart from Bragg grating demodulation), and sampling rates are likely to increase into the future. It should also be noted that, unlike the sensors themselves which are grouted inside a borehole, the CCD arrays and light sources can be easily replaced as better technology becomes available, while leaving the existing fibre optic sensing infrastructure in the ground. This research project focussed on numerical and laboratory based experimentation to determine the feasibility of fibre Bragg technology for use in deep mine environments. In future projects, actual borehole deployment of the fibre optic sensing infrastructure is desired near sensitive faults in proximity to mining activities.

Practical Implications

A new strain meter with multiple fibre Bragg grating (FBG) segments was designed. In conjunction with conventional geophysical monitoring equipment, it is expected that fibre optic sensors can provide a more detailed understanding of the deformation behaviour of underground structures.

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Selected Publications

NOTE: for a complete listing of the author's publications please go to:
https://scholar.google.ca/scholar?as_sdt=0%2C5&btnG=&hl=en&q=kaiwen+Xia

Acknowledgements

See section 3.1

5.3 Deformation Measurements in Mines

Team:
University of Waterloo – *Dr. Maurice Dusseault,* Laurentian University – *Dr. Peter Kaiser,*
University of Neuchâtel – *Dr. Benoît Valley* with *Behrad Madjdabadi*

Title: Dr. Maurice Dusseault
Role: Co-PI
Collaborators on team: Vale, NSERC, Smartec

Project Goals

This research seeks to better understand the underlying mechanisms contributing to the trigger of distant seismic events in underground mines. Hence, the main objective of the research was to study a novel deformation sensing system that is capable of monitoring deformation at large distances from mining such that a better understanding of the process leading to the occurrence of remote seismicity in mines can be developed.

Abstract

This research was triggered in part by the hypothesis that remote seismic events in mines could be triggered when gravity-driven displacements are transferred to distances far from active mining (10s to 100s of metres). Accordingly, the thesis focuses on experimental research on a novel deformation sensing sensor for future verification of this assumption.

Distributed Brillouin sensing systems (DBSs) have found growing applications in engineering and are attracting attention in the field of underground structures including mining. A test program was performed to observe DBSs response to various perturbations including axial and shear strain resulting from joint movements.

The noise level of the DBSs range was ±77 με, determined through repeated measurements on an unstrained cable. Stretching test results showed a linear correlation between the applied strain and the Brillouin frequency shift change for all strained lengths above half the spatial resolution[5] of the DBSs. However, for strained lengths shorter than half the spatial resolution, no strain response was measurable.

A borehole installation method was developed by testing the sensing cable's response while embedded in mortar beams. When the cable is directly embedded in the mortar, uncontrolled self-debonding happens that introduces uncertainties in the measurements. This limitation was overcome by anchoring debonded sections of the sensing cable at regular spacing. This arrangement produced consistent strain patterns for each strained interval.

The influence of borehole diameter and strength of the filling material were evaluated for their possible effects on the strain transfer process to the sensing cable. With the anchored arrangement of debonded cables, these properties of the grout did not have a measurable effect on the results, and as long as the tensile strength of the grout is low enough to break at the joint locations, the strain transfer performance from the rockmass to the sensing cable was excellent.

A study was devoted to understanding such a deformation monitoring system under various shear displacement conditions. These included the difference in response of the system in direct shear compared to tests performed in direct tension. The system response was evaluated for various strained lengths as well as distances over which the bending strains are acting (kink lengths). The latter was found to be an important factor influencing monitoring results. In addition, the system behaviour under shear displacement where the sensor is inclined with respect to the joint strike was evaluated to understand the effect of a combined extension and shear displacement.

Context

Typically rock failures and seismic events are directly related in space and time with active mining, generally occurring within tens of meters from the active mining front (Cook, 1976). Apart from near-field seismicity related to the mining-induced stress changes, it is expected that the more remote rock mass is seismically quiet (unless in a tectonically active area) and not affected by the mine workings. However, some failures have been observed that are spatially remote from current activity in the mine and occur at time periods not related to major blasts. These events have been considered to occur in association with geological features on a regional scale (Cook, 1976).

These remote events are often located at distances where the stress change directly attributable to the extraction of a stope would be expected to be negligible (Kaiser, Vasak, Suorineni, & Thibodeau, 2005; Urbancic & Trifu, 1995). Although stress-driven failures close to opening boundaries have been well studied through seismic monitoring (Cai, Kaiser, & Martin, 1998, 2001) and other approaches (extensometers, stress cells…), the causes for mining-induced remote seismicity remain poorly understood. Understanding of such events will increase mine safety and economy.

For this research, it is hypothesized that distant seismic events may be triggered by remote stress-changes induced by gravity-driven deformation processes (GDDP). According to this GDDP hypothesis, deformations induced by ore extraction will propagate quite far under gravitational stress. It should be noted that transfer of deformations could be facilitated by mining at lower levels hence, based on GDDP, it is a transfer of deforma-

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[5] *The minimum length of a strained event that can be accurately measured by the sensor*

tions, not stress, that leads to stress changes at remote locations, Figure 78.

This transfer process is a chain like movement of discrete, massive rock blocks. Kaiser et al. (Kaiser et al., 2005) used a train analogy to describe the sequential movement of rock blocks, in that the movement will transfer along the whole system. The last car in a train will not feel the locomotive movement immediately and will start moving only with some delay. Similarly, if there is such a delay, or a deformation gradient, in the chain-like displacement of rock blocks, stress might buildup and induce a failure if intact rock blocks and joints are already highly stressed.

The main motivation for this thesis is, therefore, to measure the deformation between two seismically active areas, i.e. old extracted zones at higher levels above the current active mining, to see if, for example, the displacement rate at point Ai is similar to that at point A1 (similar to the condition where particles can flow in a silo), or to examine whether rock blocks are moving at different rates. To this end, measurements with a distributed deformation sensor that can return many measurement points along its length will enable us to study the deformation field away from active mining.

Methodology

Such a distributed deformation sensor was installed in the Coleman Mine in Sudbury for the first time. A 25 m high sill pillar was instrumented with five distributed sensors and monitored for over two months. Three sensors were broken in early stages of the sill pillar mining, and analysis of measurements from the two remaining sensors did not yield much information. On one of these two sensors, a very high strain response was noticed in a small section, a response quite different from the rest of the sensor data. This high local strain was found to be associated with a rock block detaching from the mine back at a joint location, and confirmed with the results from optical imaging of the borehole. It was then decided to study the strain response of such a system in great detail in the laboratory before trying more field installations.

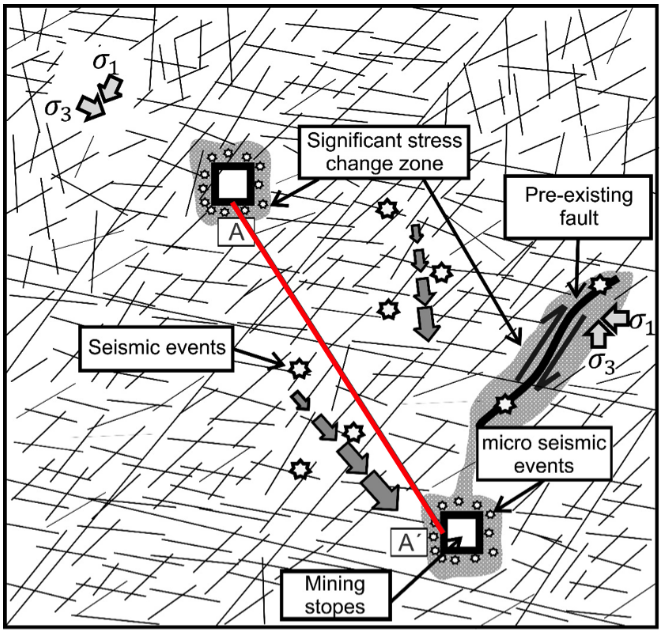

Figure 78: Remote seismicity around seismically active area A when mining is happening at A

A better understanding of the relation between deformation fields within the rock mass and remote seismic events can help test hypotheses such as at remote distances. The tested deformation measurement device is therefore meant to be deployed to record the deformation pattern away from active mining. Although optical fibre technology has been in use for almost a decade in different engineering contexts, there is a limited number of cases where such a system has been applied in underground mining. Consequently, a detailed examination of a suitable fibre optic system is needed before deployment in an underground mining environment where many activities and factors may have an adverse effect on measurements.

This research presents the findings of a comprehensive laboratory testing program that was set up to explore the following:

- study the performance of the selected system in terms of accuracy, resolution, and repeatability;
- define the base noise level of the system;
- establish the relation between deformation and the system's internal measured parameter, i.e. frequency;
- explore the sensing system's response to direct tension;
- explore the system's response to spatial and sequential distribution of applied deformations;
- investigate the strain transfer processes when the sensor is embedded in a filling material;
- explore time dependencies of the system response when under constant strain; and
- investigate the response to direct shear displacement when the sensor is installed in a borehole crossing various joints at different inclinations.

In addition, practical questions for successful utilization of fiber optics are addressed:

- Is the system capable of registering the relative movement of a series of rock blocks next to each other?
- What would the system response be when rock blocks deform in different sequences over the course of mining?
- What is the best installation procedure?
- What grout properties are required for hole filling to achieve optimal strain transfer?
- What borehole size should be used for the system to have optimum performance?

Summary of Findings

Continuous interrogation of an unstrained SMARTprofile cable was performed to characterise the noise level below which no changes in temperature or deformation could be detected. Also, a spatial shift along the optical cable was noticed from various overlain strain profiles, and this was partially corrected using a comparative algorithm. The results show that the system can provide high performance in strain sensing with a minimum noise level of ±75 με.

Extensional tests on various strained lengths show that the strain could be accurately detected down to a strained length above half the claimed spatial resolution, i.e. 25 cm, Figure 79 (a) and (c). Below this length, the registered response is unreliable, highlighting a system limitation for the measurement of short strained length sections. This limit also dictates the base requirement for future test designs and field installations (e.g. clamping points spacing). During these tests the frequency strain conversion factor was re evaluated and refined, Figure 79 (b).

The behaviour of deformation events over a strained length shorter than LSR/2 is better understood when considering Brillouin gain spectra. It was found that ΔvB changes are not linked to the strained length, but the peak gain can be correlated with the strained length.

The installation method for the current objective of DiTeSt usage is borehole placement with grout filling. A major part of the experiments was to study the interaction between an embedded cable and the mortar beams (i.e. grout) and the effect of mortar properties on the strain transfer process.

When the cable was fully embedded in and bonded to the mortar beam, the DiTeSt response was different at each of three 55 cm spaced induced crack locations. This was found to be significantly influenced by the self-debonding length evolution with applied displacement. Apart from the positive point that the system could register frequency changes during cable debonding, the amount of crack opening could not be estimated with high accuracy since the strained length was not constant. The same test implemented with weaker mortar showed that the debonding process then became more uncontrolled.

The cable was then anchored at 50 cm spacing and pre debonded from the mortar between anchors. Inducing one crack over each anchored interval resulted in a totally different frequency compared to that in the fully bonded cable test. The frequency change registered over each interval was quite repeatable and consistent with the applied displacement since the strained length was constant, Figure 80.

Finally, the effect of filling thickness and strength on the strain measurement effectiveness were evaluated using a larger diameter beam and a beam formed with weaker strength mix, respectively. In both tests, DiTeSt showed exactly the same performance as with a smaller beam and stronger mortar. This is related directly to the cable embedment method in the mortar. Therefore, the most important characteristics of the filling material should be a low tensile strength so that it breaks at the joint locations as soon as they start moving.

The best method of installation based on the experimental studies on the sensing cable under pure tension should be the anchoring debonding method with minimum 50 cm anchor spacing and a method to give high quality and complete debonding between anchors.

The test involving various strained lengths (l) from one to 100 cm showed that all lengths above 30 cm could show a frequency response over the strained length, although sometimes no response was observed for LSR/2≤l≤40 cm (LSR is spatial resolution of the DiTeSt). Furthermore, it was noticed that the Brillouin frequency shift change (ΔvB) varied in a parabolic trend with shear displacement, contrary to axial tests where a clear linear correlation was established, Figure 81. Apart from the strained length, another important factor was taken into account for its effect on

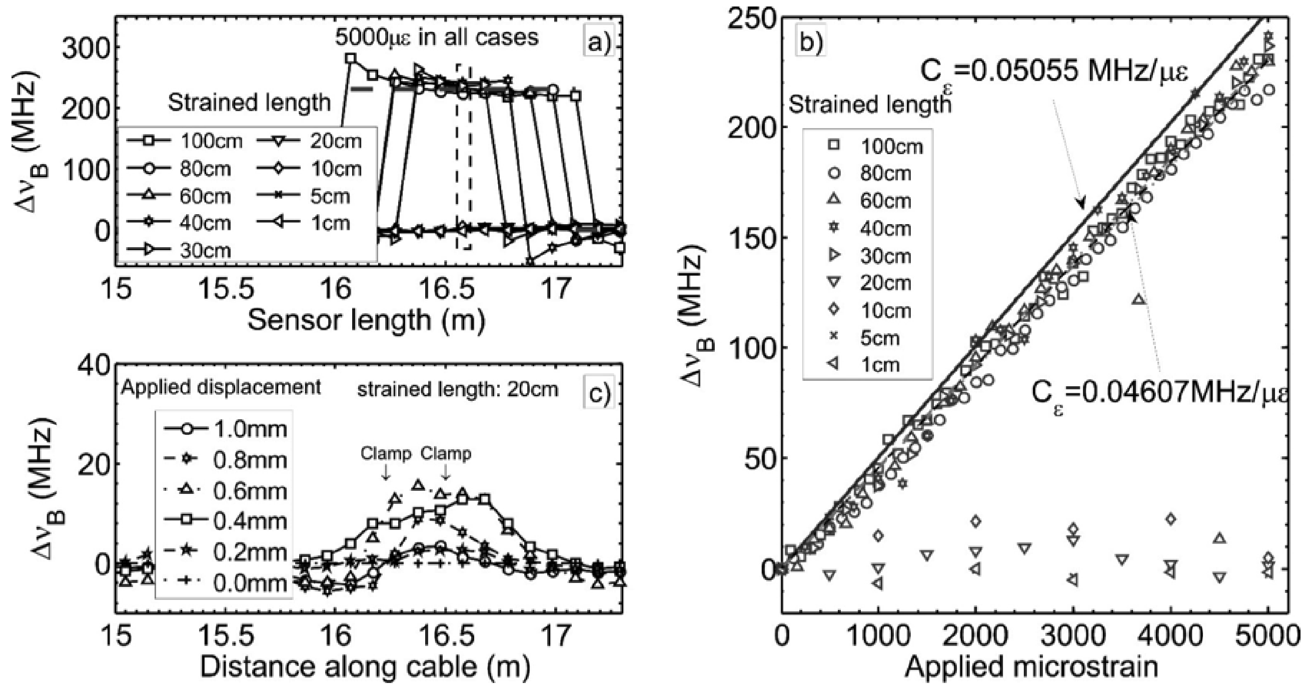

Figure 79: a) Brillouin frequency shift change (ΔvB) for application of 5000 με to various strained lengths b) best-fit correlation between applied strain and ΔvB c) typical ΔvB response applying incremental displacement to a 20cm clamped length

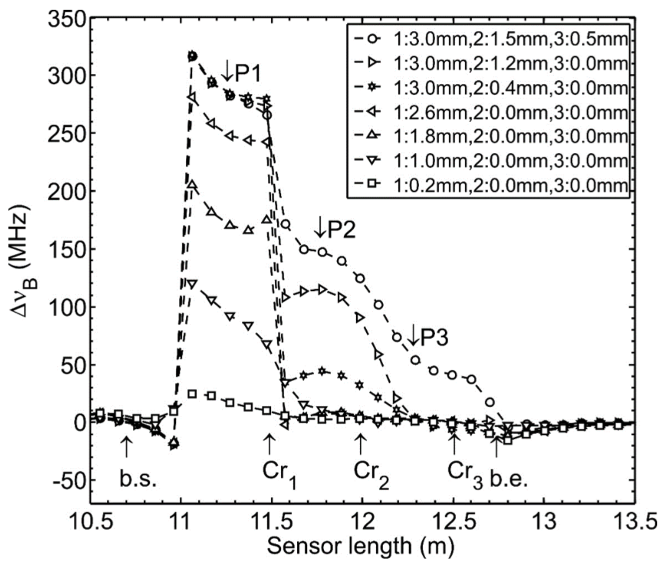

Figure 80: ΔvB for the debonded cable embedment method. The full gage length is mobilized at the early displacement stage. P1-P3, Cr1-Cr3, and b.s., b.e. represent three points in the middle of each strained section, three cracks, beam start and end locations, respectively

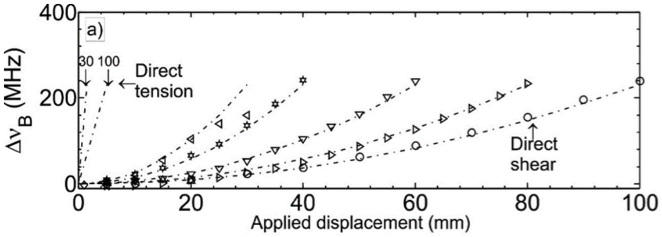

Figure 81: ΔvB changes in parabolic trend with shear displacement in comparison with the linear correlation from direct tension tests

the shear displacement response of the system. Reducing the kink length (LK), a length over which the transverse effect of the shear displacement occurs, showed a significant increase in the frequency response. The frequency profile also showed a slightly higher response where the kink guides were located along the strained length, particularly noticeable for shorter kink and longer strained lengths.

Another influential parameter on the system behaviour was the relative joint cable inclination. A section of SMARTprofile was laid out on the test setup at 75° from the joint trace and sheared in a way to induce elongation in the sensor. The ΔvB DS showed that the frequency still followed the parabolic trend observed in direct shear (90°) but with less intensity in curvature and a clear linear trend. The linear trend was due to the longitudinal component of the shear displacement that caused an axial strain in the sensing cable. Furthermore, ΔvB from all kink lengths showed less divergence with shear displacement relative to large differences for the same kink lengths in direct 90° shear.

The effect of the presence of more than one joint over the strained length on the system response was tested whether the cable crossed the joints at 90° or with an inclination. From the 90° shear test results it was found that a given ΔvB profile could be obtained through various displacement paths. Also, it was found that the displacement at each joint should be individually incorporated into total displacement in order to understand ΔvB DS response. For the shear with an inclined cable with respect to the joint trace, it was noticed that the frequency profile became non uniform when the cable at one joint was under elongation and at the other joint was under shortening during shear displacement. The frequency profile showed higher values in elongation, and dropped to smaller values at locations closer to shortening displacement.

Commercialization

There is an interesting opportunity for a start-up company that provides the instrumentation and monitoring services and produces great knowledge of large complex surficial and underground structures generally in geotechnical fields and specifically in the mining industry using such sensors.

Practical Implications

These sensors, if properly located in an underground mine, can provide a very good picture of the displacement field around active mining and its evolution with mining progress.

Having these data, one can notice where, why, and how displacement trends are growing and potentially correlate with existing or progressively forming structural features within the rockmass. Such data can help to modify underground support system designs or extraction sequences to provide a more stable working environment and more economically-wise decisions for mine production.

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Madjdabadi B, Valley B, Dusseault MB, Kaiser PK. Experimental evaluation of a distributed Brillouin sensing system for measuring extensional and shear deformation in rock. Measurement 2016;77:p.54–66.

Madjdabadi B, Valley B, Dusseault MB, Kaiser PK. Experimental evaluation of a distributed Brillouin sensing system for detection of relative movement of rock blocks in underground mining. Int. J Rock Mech. Min. Sci. Submitted-Under review.

Madjdabadi B, Valley B, Dusseault MB, Kaiser PK. Experimental evaluation of a distributed Brillouin Sensing system under shear displacement condition. Under preparation.

NOTE: for a complete listing of the principal author's publications please go to:
https://www.researchgate.net/profile/Maurice_Dusseault

5.4 Underground Mine Construction – Ground Support for Constructability in Highly Stressed, Brittle Failing Rock Masses

Team:

Laurentian University and CEMI team – *Dr. Peter.K. Kaiser* and many collaborators worldwide,
sponsored by Rio Tinto at the Rio Tinto Centre for Underground Mine Construction at CEMI

Context

Research in support of underground construction was the mandate of the Rio Tinto Centre for Underground Mine Construction at CEMI. However, several aspects were also supported by the SUMIT Program.

In 2016, Dr. P. K. Kaiser was invited by the World Tunnelling Association to deliver the Muir Wood lecture in San Francisco. This provided an opportunity not only to draw on his experience in deep mining and Alpine tunnelling where both static and dynamic failure processes have to be managed as mining-induced stresses cause shallow and deep-seated rock mass failure but also to summarize research findings at Laurentian University conducted as part of the SUMIT Program. Based on the author's geomechanics background, the lecture focuses on geotechnical design aspects. Findings from collaborative research with colleagues such as Drs. M. Cai (Laurentian Uni.), E. Eberhardt (Univ. of British Columbia), M. Diederichs (Queens Univ.), E. Hoek, D. Martin (Univ. of Alberta) and many more were reflected in the lecture and in Kaiser (2016).

Executive Summary

For the economic and safe construction of deep tunnels, a contractor must be presented with efficient and effective ground control measures, i.e., support classes that can be rapidly installed and are effective in managing stress-fractured ground. For this purpose, it is necessary to properly anticipate the rock mass behaviour and then provide a flexible but reliable means for the support of a shell of stress-damaged ground around the excavation such that a tunnelling project can proceed without unnecessary delays. Stress-driven rock mass failure in brittle rock in the form of gradual ravelling or sudden strainbursting may often slow the tunnelling progress. Both failure processes impose difficult and hazardous conditions for tunnel construction whether the tunnel is advanced by TBM or by drilling and blasting.

Robust engineering integrating empirical experience, engineering analysis and sound construction methods provide the key to successful tunnelling and timely project completion. Designs that respect the complexity and variability of the ground, consider the practicality and efficiency of construction, and ensure the effectiveness of ground control measures, lead to fewer project interruptions and, consequently, fewer claims or cost overruns. Robust engineering in highly stressed, brittle failing rock requires a design which takes rock mass degradation into consideration and ensures that all construction tools work well with broken rock.

Some challenges to overcome for economic constructability are engineering design aspects that matter for the selection of the most appropriate excavation technique and the most efficient and effective support systems. This article discusses both fundamental and practical means to better understand and properly document (e.g., in tender documents) the anticipated rock mass and excavation behaviour; better qualify relevant rock mass characteristics and their variability; adopt representative models and inputs to capture the actual rock mass behaviour; and account for practical constraints during construction (i.e., by matching a design to a chosen construction technique).

These elements of excavation design are discussed and the impact of naturally variable conditions is reviewed. Guidance is provided for the quantification of anticipated rock mass behaviour near excavations and for the selection of design inputs to arrive at ground control measures that provide both safe and efficient construction. It is discussed how stress-fractured ground is prone to ravelling and how it imposes large deformations due to rock mass bulking. It is concluded that deformation-based support selection principles are most suitable for conditions with stress-fractured ground. Guidance is also provided to arrive at support systems that are suitable for highly stressed tunnels in brittle rock for civil construction and mining operations.

The primary conclusions highlight the need for improvements in better anticipating the rock mass behaviour at the tender stage and the need to design ground control measures from a perspective of practicality rather than theoretical analysis.

Specifically, with respect to excavation stability assessment, it is necessary to anticipate brittle failure processes and to properly describe the implications of shallow rock mass damage and bulking of stress-fractured ground (e.g., stand-up time reductions).

With respect to support selection, conventional support design approaches, based on standard rock mass rating systems, are severely limited in conditions where stress-driven failure processes dominate. They do not provide effective support systems for stress-fractured ground, because they do not account for mining-induced stresses, stress changes and related deformations. For tunnelling and mining at depth, it is necessary to select support systems that are effective in controlling the bulking of broken rock and able to yield when strained by deformations imposed by the stress-fractured ground. This can be achieved by following the deformation-based approach described in this article and by Kaiser (2014). Since rock mass bulking can impose excessive deformations that cannot be estimated by standard numerical models, it is necessary to separately estimate the deformation demand when bulking dominates (e.g., in late stage mining).

With respect to constructability, it is concluded that conditions of brittle failure must be anticipated early and thus well described in a quantitative manner in tender documents. This should include design inputs relevant for estimating stress-fracturing, for anticipating the extent of rock mass degradation and its impact on stand-up time. Most importantly, flexible and effective support systems (classes) must be provided to manage rapidly changing ground conditions and prevent related delays.

Selected Publications

Kaiser PK 2016. Ground Support for Constructability of Deep Underground Excavations – Challenges of managing highly stressed ground in civil and mining projects. Sir Muir Wood lecture of International Tunnelling Association at World Tunnelling Congress, San Francisco, 33p.

Kaiser PK 2014. Deformation-based support selection for tunnels in strain-burstprone ground. DeepMining'14, ACG (eds. Hudyma and Potvin), p.227-240.

5.5 Pillar loading projects

5.5.1 PILLAR LOAD SIMULATION

Team:

Laurentian University and MIRARCO team – *Drs. P.K. Kaiser, C. Gonzalez* and *M. Cai*

Context

As mines progress to great depth and more low grade ore bodies are converted to caving operations to facilitate mass mining, designing stable pillars with effectively supported stress-fractured rock using deformable support systems becomes a necessity. Caving allows for bulk mining of marginal and large ore bodies and is practiced by mining companies in Canada and around the world.

Experience shows that support installed in highly stressed hard rock pillars is often ineffective because the support system cannot manage the large displacements imposed by fractured rock. This is due to a lack of information regarding pillar loading and rockmass behaviour when hour-glassing occurs. Critical design factors include: pillar loading as a function of mine sequencing and loading by fragmented rock during ore draw, stress-driven brittle rockmass failure and its influence on support behaviour. This project aims at developing effective tools and techniques to improve pillar design and pillar performance. Specifically, it focuses on two pressing tasks:

- Time-dependent pillar loading: investigating processes resulting in pillar loads from compaction of fragmented rock above pillars (in drawbells or draw column); and
- Deformation-based support selection: matching deformation demand, resulting from the above described pillar wall fracturing process, with the support's yield capacity.

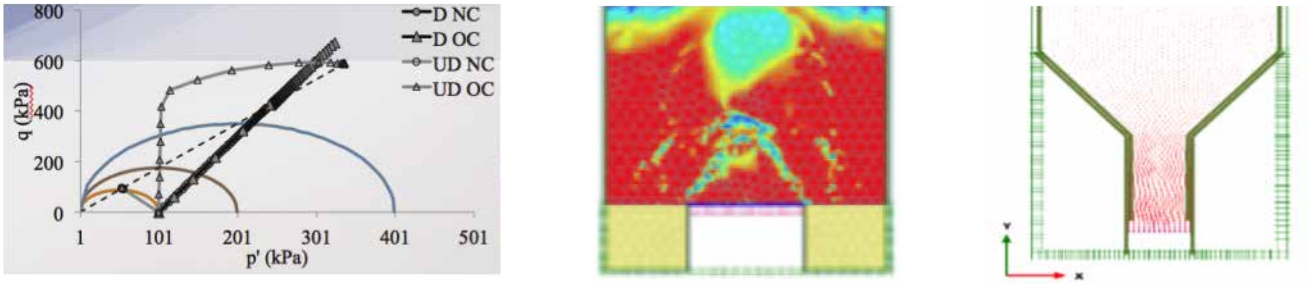
This project is still ongoing with leverage funding from the NSERC CRD program.

Some of the findings from the second task are reported elsewhere in this SUMIT report. Here we focus on the first task, in particular on modelling pillar loading with critical state material behaviour models to evaluate the impact of dilation and compaction of fragmented rock on pillar loads.

Effective rock support design and draw control can only be achieved with an improved understanding of the pillar loading process. As indicated above, a system of pillars in an active mining operation is susceptible to a variety of stressors. This research addresses some of the most important influence factors by use of laboratory measurements (conducted at the University of Chile with alternate funding) and advanced numerical modelling using critical state models.

Methodology

After stoppages have taken place in drawing ore, time-dependent compaction of caved rocks has been observed and this time-de-



(a) (b) (c)

Figure 82: (a) Critical state concept in p'-q space; (b) trap door analogue for pillar loading showing strain localizations; and (c) silo analogue for drawpoints in caving operations

pendent behaviour of fragmented rock can alter the stress distribution in and on pillars (on draw bell walls) and impact pillar loads when drawing resumes. In this study, we investigated two of many influence factors:

- time-dependent grain size distribution changes: point-loading between fragments causes stress raisers that with time will lead to tensile fracturing of fragments causing a shift in the grain size distribution curves to smaller particles; and
- time-dependent compaction or dilation: from critical state soil mechanics principles applied to fractured rock, called rockfill, it is understood that fragmented rock will dilate or compact during ore flow. As a result, the fill changes its strength, may cause stress arching and transfer higher loads to the rockfill/pillar interfaces. This process has been studied for silo design but has not yet been applied to pillar loading in mines.

PFC models are used to simulate the load- and time-dependent fragmentation (due to stress-corrosion) and related compaction process.

The second process is captured in Figure 82 by the critical state models (a) and the simulation of load transfer to pillars using a "trap door" models (b) or silo models (c).

Critical state modelling work was first initiated with a Post-doctoral Fellow in collaboration with the University of Catalonia using the Plaxis code and then continued at MIRARCO using RS2. Experimental studies on rockfill materials for dam construction provided insight into the response of fragmented rock to stress change and some of the published results were used to evaluate the numerical model results (not reported here).

First, the computer codes Plaxis and RS2 were compared and found to produce equivalent results even though some of the modelling parameters differ. As is typical for critical stage analyses, the results are presented in the p'–q space (Figure 82 a), where:

$$p' = \frac{\sigma_1' + \sigma_2' + \sigma_3'}{3}$$ is the effective mean stress;

$$q = \sqrt{\frac{(\sigma_1' - \sigma_2')^2 + (\sigma_2' - \sigma_3')^2 + (\sigma_1' - \sigma_3')^2}{2}}$$ is the effective deviatoric stress; and

and the yield surface is defined by a logarithmic expression:

$$q = Mp' \ln\left(\frac{p_0'}{p'}\right)$$

where, p'_0 is the hardening parameter that defines the yield surface size and M is the slope of the critical state line (CSL). The CSL separates compacting (to the right) from dilating (to the left) material behaviour modes. The state to the right is called "wet" because pore pressures would be generated if saturated. The state on the left is called "dry" because of the dilative behaviour leading to pore pressure reductions.

Summary of Findings

Here we summarize the current state of the modelling for the second task. The eventual outcome of the research will include pillar load evolution charts, which should assist in selecting pillar design loads and in managing draw control.

Single drawpoint model

A set of stress paths in a p-q diagram are graphed in Figure 83 for five points located along the vertical axis of a single drawpoint model (for the simulated dry conditions, p' = p). The CSL is represented by the line intersecting the origin.

The initial p-q values are given by the original compaction due to the gravitational forces. In this particular model, only one point, the point closest to the trap door simulating ore draw, reached the critical state; the others are tending toward the CSL. This is because the points further from the drawpoint experience internal stress redistributions as anticipated by the silo theory. As the support pressure at the drawpoint decreases, the broken rock deforms until it reaches its critical state and starts to yield and flow. The stress and deformation distribution is not uniform and internal stress arches may develop. Figure 84 shows the same family of curves together with the respective yield surfaces (ellipses).

Because stress paths on the right of the CSL indicate contraction or compaction, the fractured rock is compacting at all locations; i.e., the rockfill is compacting at all locations with the chosen material properties. Furthermore, all stress paths indicate unloading situations. For wet muck, pore pressures could be generated in a fully saturated rock fill and effective stress analyses are required to fully capture the behaviour of saturated rockfill.

Double draw point model

Figure 85 presents a double draw point with different fill properties on either side and hardening parameters p'_0 = 1.2 and 7.3 MPa,

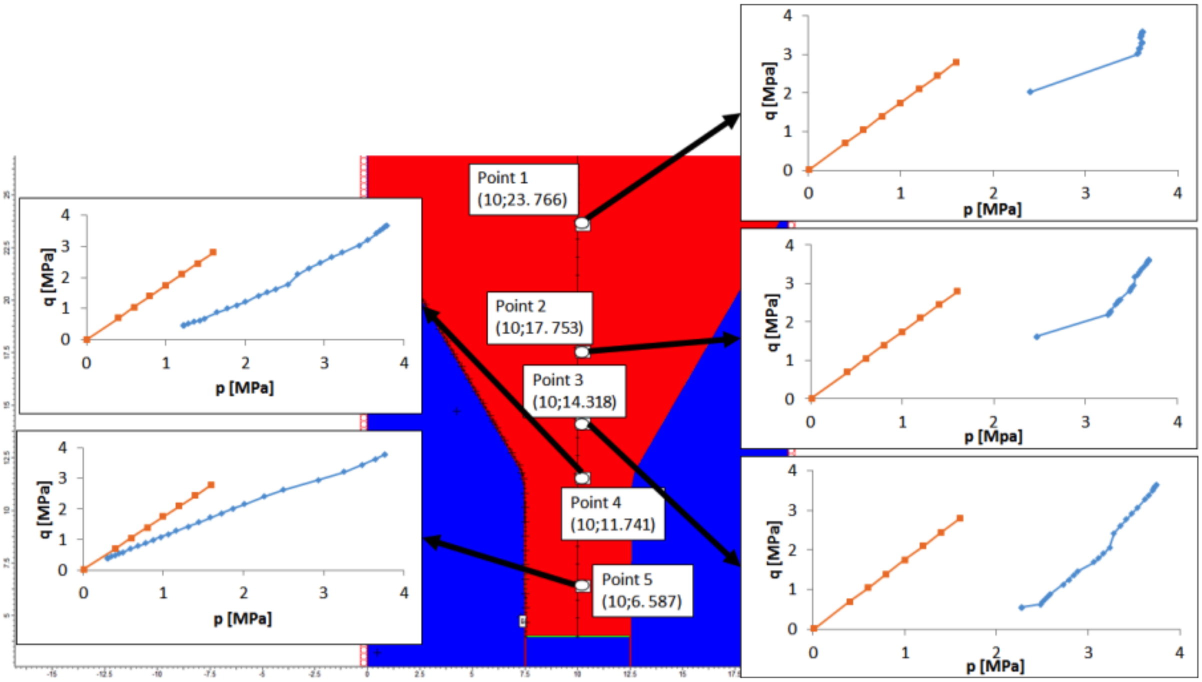


Figure 83: Stress path in mean stress p vs. deviator stress q space at five locations

respectively. The initial stress state was defined by a low earth pressure coefficient to rest k_0 = 0.4 leading to a relatively high deviator stress q. A higher p'_0 reflects a denser material; in this example the material on the right (blue) is denser than on the left (pink). Draw is simulated by simultaneous "trap door" movements. Hence, the material on the left, which is weaker and reaches the yield point more rapidly deforms more easily.

Here we present results from simulations of loose ore in dual draw points. The volumetric strain is shown in Figure 86 together with yielding elements and normal stress distribution at the joints between the ore and pillars at an moderate draw stage.

The volumetric strain patterns and therefore the extent of material movement zones are strongly affected by the hardening parameters (or the initial density of the broken rock). The looser fractured rock I (on the left) reached the critical state in the centre (+) due to its own weight and later near the contact to the pillar due to the stress redistribution. Hence, the volumetric strain contours are localized near the transition to the vertical pillar wall. The denser fractured rock (on the right) reached the critical state only near to the transition from the inclined to the vertical pillar wall, i.e., the ore in the right draw point does not yet yield.

The normal stress distribution shown in black at the interface between ore and pillar (simulated by a joint) is uniform on the inclined surfaces and much lower on the vertical pillar walls. Hence, the pillars are essentially unconfined on the vertical surfaces while being loaded on the inclined surfaces.

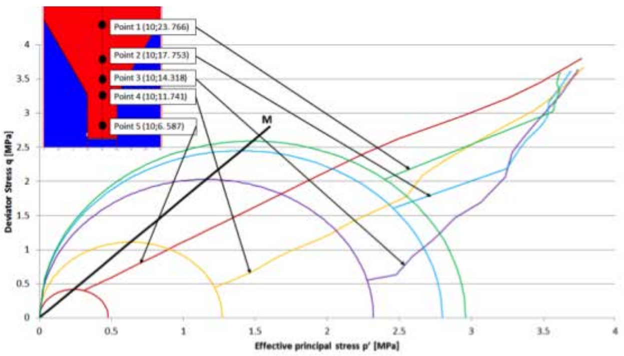


Figure 84: Stress path in p–q space for the five locations on the vertical axis (x and y coordinates are given in brackets)

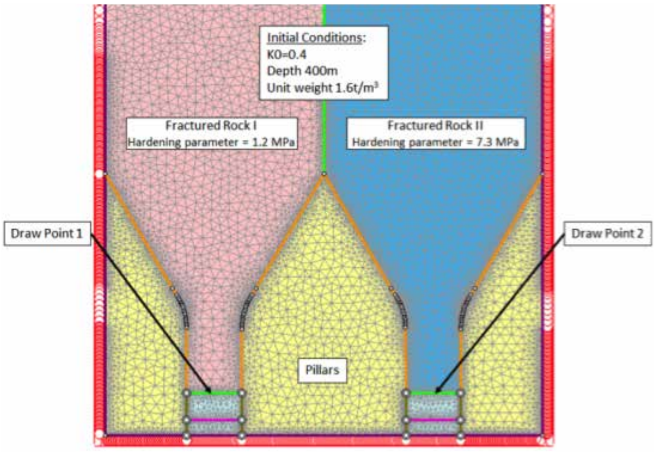


Figure 85: Double draw point model with denser material on drawpoint on the right

Ore behaviour in single draw point

The rockfill is again assumed to be normally consolidated. Hence, it is initially on the wet side and in a contractive state. Figure 87 shows the initial state because of self-weight. This loading tends to compact the material and, due to the pillar geometry, the compaction is not uniform and part of the material yields as shown the Figure 86 by (+). The stress state of the rockfill that yields is located outside of the yield surface (in Zone 3; at the large ellipse), whereas the rock fill that is compacting but does not yield remains inside the yield surface and is in an elastic state (Zone 1). Finally, rockfill that is dense enough so it will dilate when deformed would be located on the left side of the CSL, i.e., in the dilating space (Zone 2; not reached with the chosen parameters in this model).

As the "trap door" is lowered to simulate ore draw, the rockfill changes its characteristics and transitions from one zone to another. This can be interpreted in terms of stress paths in the p-q space as shown in Figure 88.

The critical state line is defined by the slope M, and it is typically obtained based on triaxial compression tests. However, in a silo situation decompression dominates the material behaviour and it is more appropriate to obtain the M parameter from triaxial extension tests. The CSL line accounting for the dependence on degradation can be obtained by a correction factor K such that q = K M p'. The critical state slope, modified by the parameter K, is flatter as illustrated by M' in Figure 88.

In the above presented drawpoint simulation, the initial rockfill density was selected to fall very close to the CSL (defined by M') and some of the stress paths seek to reach the CSL as material is drawn from drawpoint (trap door is lowered). These points are initially located in Zone 3. However, points initially located in Zone 2 change their behaviour by moving towards Zone 1 (elastic) and show a stress path of increasing mean stress p whereas the deviator stress remains constant. It thus tends to a stable condition when reaching Zone 1. This creates a stress arch in this area. This stress arch is in the contractive behaviour zone (Zone 1 is elastic and contractive) but it is able to transfer loads to the pillars.

Figure 89a presents the volumetric strain distribution at a stage of draw when the pressure on the "trap door" has reached a minimum. Only contours of positive values of volumetric strain are shown in order to highlight the arch with near zero volumetric deformation.

According to the silo theory, this arch is in compression not in extension. Thus, points initially located in Zone 2 start with negative volumetric deformation (contraction) as the arch is developing. However, as can be seen in Figure 89, the points at the top and at the bottom of this region, initially located in Zone 2, stay with zero volumetric deformation and follow an arch shape distribution. On the other hand, near the pillar walls, the compressive stress increases and points initially located in Zone 2 become elastic (in Zone 1) until they yield under compression (Zone 3).

The critical state identified by dark pluses (+) indicates points that start yielding (in general in Zone 3) due to the self-weight, and the red bright coloured pluses (+) identify locations that reach the critical state and yield by arching. This is because the arch compresses towards the pillars and the material close to the pillar wall yields under compression (Zone 3). This compressed material is located along the inclined pillar walls.

The volumetric strain distribution in Figure 89a is representative of points that have reached the critical state. The horizontal displacements towards the inclined pillar walls takes place as the stress redistribution develops for points initially located in Zone 2. The corresponding horizontal displacements are shown in Figure 89b.

Pillar loading in single draw point

The stress evolution on the pillar walls for various material characteristics and status relative to the critical state is currently being investigated. An example is presented in the principal stress space (Figure 90) for two locations: at the apex of the pillar and at the vertical pillar wall. Both points are at the edge of the arch with zero or almost zero volumetric deformation. The contours shown bound the volumetric deformation, ranging between values closer to zero, i.e., where yield develops along the inclined pillar wall.

At or near the apex, the minor and major principal stresses increase in part because of the arching process described above. This means that the pillar load is gradually increasing as ore is drawn and arching develops. The maximum pillar load is experienced when the arch is completely developed and ore flux is stopped above the arch.

On the other hand, at the vertical wall, the minor principal stress initially decreases at a more or less constant major principal

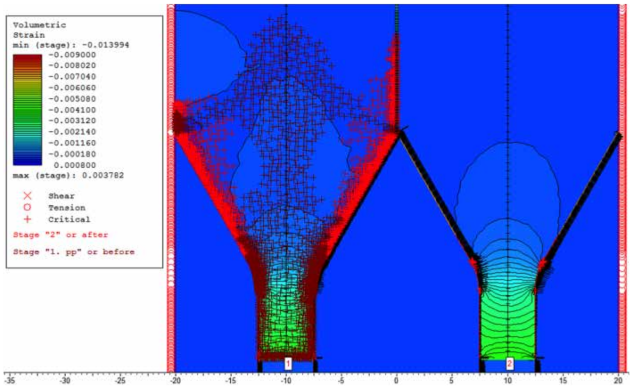


Figure 86: Volumetric strain contours and yield elements (x shear, and + at critical state), as well as normal stress distribution at joints (in black)

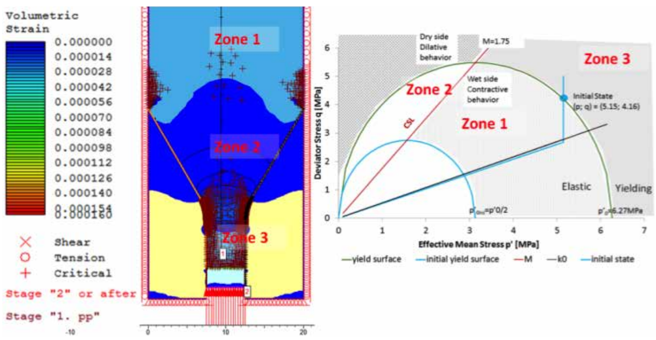


Figure 87: Initial volumetric strain associated to the initial stress conditions due to self- weight of fractured rock

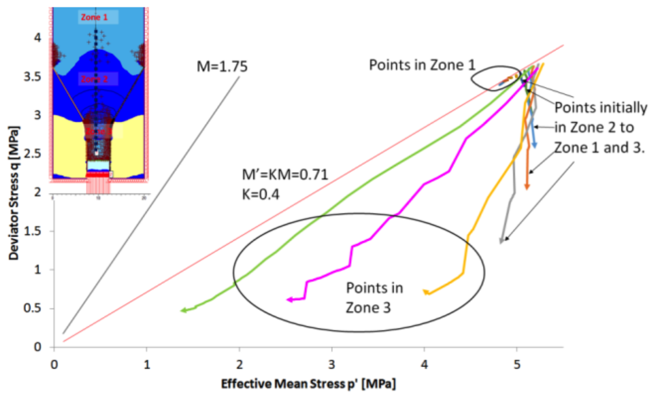


Figure 88: Stress paths at various locations along the points located in the middle of the model

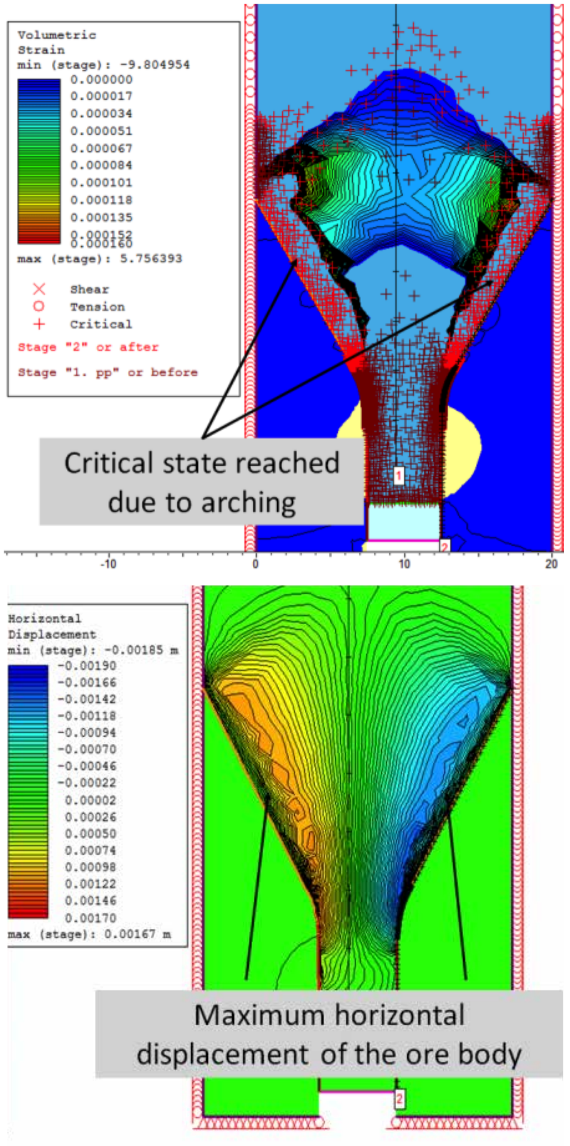


Figure 89: (a – top) Volumetric strain and (b – bottom) horizontal displacement contours

stress. As a consequence, the confinement of the pillar drops. At higher draw displacements though the major principal stress increases and this means that the shear stresses at the wall eventually increase.

Futures studies will investigate the influence of various initial material properties and draw sequences and rates on pillar loads and pillar confinement such that the pillar failure processes can be better assessed. It is the intent of this investigation to establish pillar loading charts for this purpose.

Conclusions

This investigation shows that RS2 offers a tool to investigate the impact of fractured rock on pillar loading using critical state models as adopted for rockfill in dam construction.

The impact of various hardening parameters, and over-consolidation ratios, OCR, the initial earth pressure coefficient at rest k_o can be investigated. While not yet tested, the fluid flow capability of RS2 will also allow to assess the influence of pore pressure changes.

It is evident from this preliminary study that factors that influence the fractured rock density (time dependent fragmentation and compaction) and the relative movement between drawpoints affects pillar loads. The critical state models are well suited to establish pillar load charts.

Future work will focus on the calibration of models on results from sand flow tests and eventually by back-analyses from pillar monitoring. The influence of arching and unloading as well as the consequences for pillar stability and support design can then be better assessed.

Practical implications

This project addressed pillar loading for optimal support selection and pillar design. The potential cost-benefits of this research are significant. Such benefits will not result from the proposed research alone but will be derived by mining companies applying the tools, charts and models developed in this project. The impact will be seen at green fields projects when designing layouts (pillar dimensions), during mine development (when selecting effective and efficient support), and during operations when controlling draw in caving operations). With improved understanding of pillar loading and time-dependent behaviour of fragmented rock, sound engineering can be conducted, and this will improve safety and productivity in underground mines.

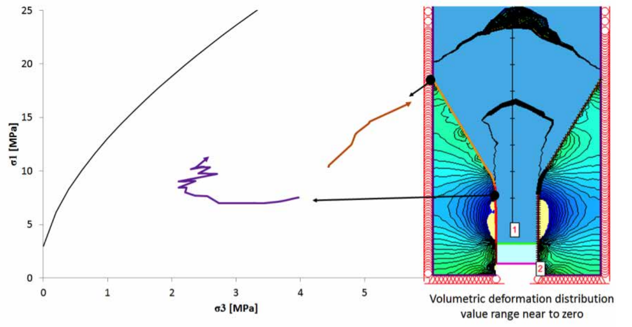


Figure 90: Stress paths at two points at the pillar to rockfill interface where the volumetric deformation contours is near zero

5.5.2 NRS PILLAR MONITORING – CHARACTERIZATION & TECHNOLOGY TRIALS

Team:

Laurentian University and CEMI team – *Drs. P.K. Kaiser, Salina Yong* and *D. Duff*

Context

This project was motivated by the potential benefit of higher footprint reliability and the observations of pillar instabilities at many operations. For better support of footprints and selection of risk-balanced pillar dimensions, it is necessary to better understand the behaviour of the brittle failing rock near the excavation wall by estimating the depth of failure and rock mass bulking. Reliable detection of depth of wall damage is urgently needed for four primary reasons:

- confirmation of damage depth estimation based on rock mass strength (in situ and support) combined with situational loads;
- assessment of reliability and effectiveness of installed support systems;
- development of innovative support systems (gabion concept); and
- support system integrity assessment.

For this study, Sudbury Integrated Nickel Operations, a Glencore Company's Nickel Rim South (NRS) Mine provided a test site to evaluate geophysical techniques for depth of failure monitoring and remote sensing techniques for monitoring rock mass bulking. The monitoring program was expanded to address the parallel interests of the NRS Mine for strainburst management and assessing the conditions of narrow rib pillars and failure processes.

The objectives of the field trials were:

- damage characterization of a pillar in response to nearby mining;
- assessment of the pillar's response to further mining-induced loading;
- determination of the depth of failure behind the support;
- testing of seismic refraction tomography to define the depth of failure; and
- testing of a hand-held laser scanner (ZEB1) for rapid detection of rock mass bulking.

Assessments of the chosen technologies required verification by conventional means; thus, secondary goals included characterization of the pillar and monitoring its response as two panels were stoped. Ground-truthing methods included core logging, laboratory testing for UCS and P-wave velocities, borehole camera logging, borehole velocity profiling, stress change monitoring, MPBX deformation monitoring, and convergence measurements. The field trial spanned eight months and involved 20 key personnel and five organisations (including Sudbury Integrated Nickel Operations, a Glencore Company and the Rio Tinto interns).

Summary of Findings

The outcomes and practical implications of the field trials are:

Damage characterisation of the pillar was successfully achieved by integrating the various datasets (Figure 91). Velocity-profiling through the width of the pillar indicated stress damage throughout the pillar. However, a narrow load-bearing core (0.5 to 1 m wide) in the 5- to 7-m wide rib pillar was delineated from observations of borehole breakouts and discing in drillcore. This is consistent with the minor seismicity observed during borehole drilling.

The pillar response, determined from stress change and deformation monitoring, indicated that the mining of the panels added some 100 MPa to the vertical loading and change in the bulking factor was small (<1%).

The prevalence of stress damage through the narrow width of a rib pillar presented a significant challenge to the use of refrac-

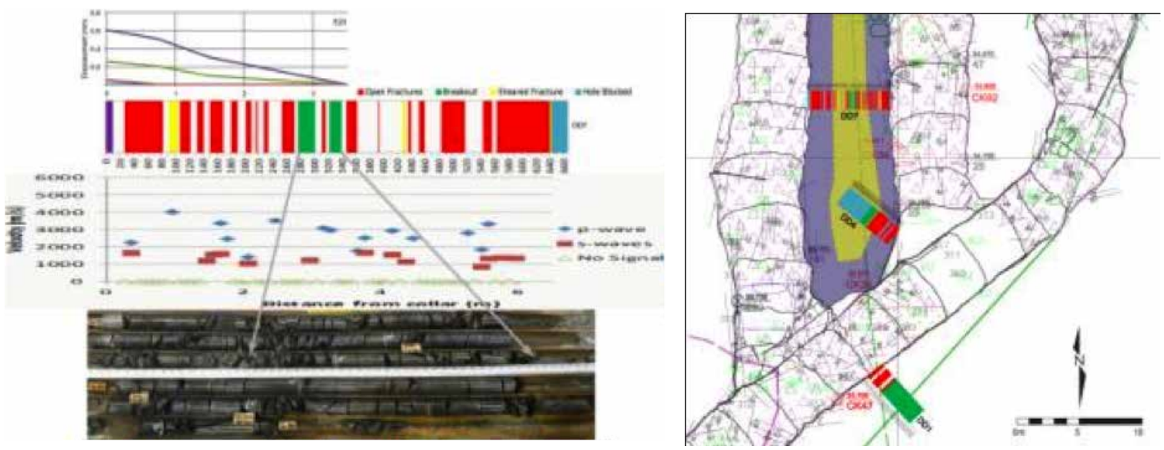


Figure 91: Integration of pillar borehole data (left) consisting of core photo, velocity profile, borehole camera, MPBX deformation profile (from bottom to top) and plan view of zones of fracturing and breakouts (borehole overlays in right) from borehole camera compared to drill core disking (underlying two-colour zonation in right)

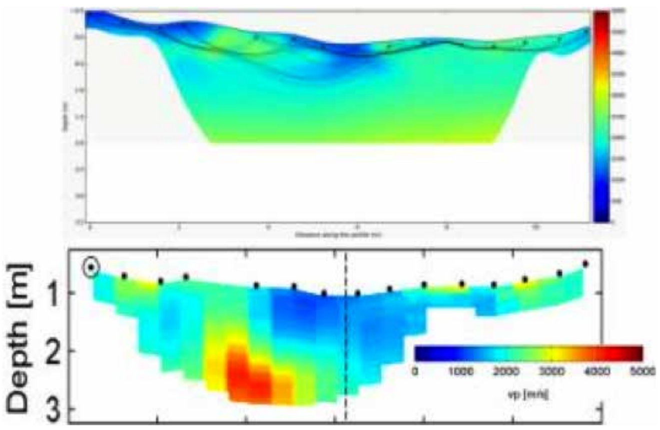


Figure 92: Horizontal seismic refraction tomograms for the pillar wall from two consultants (Golder Associates in top and Augeos in bottom, with identical gradient scales)

tion seismic for defining the depth of failure (Figure 92). The poor rock quality near the surface resulted in strong attenuation of the high-frequency energy of the seismic waves in the first 0.2 to 0.3 m. The technology is not yet ready for deployment for this purpose and recommendations include deeper seating of the sensors, decreasing their spacing, and using advanced processing methods to improve data quality.

Field testing of the handheld laser scanner showed promise in its ease of use and data processing. The limitation of the system is set by its accuracy, which was found to be ±5 mm. Comparison against convergence measurements (Figure 93) suggests that the ZEB1 scanner can be a viable alternative for convergence monitoring for situations where deformations clearly exceed the system accuracy.

From NRS's perspective, this study was primarily undertaken to identify stability indicators for the assessment of pillar load capacity for dimensioning and rockburst prevention. The load-bearing core of limited extent is still able to store strain energy and thus, is burst-prone to some extent: it is able to generate seismic events and release some stored strain energy. However, because of the highly-damaged shell in the walls on either side, any energy release will be dampened in the low velocity zone and dynamic deformation will be cushioned.

This limited stored energy is not expected to impose excessive deformations or stresses on the support. Considering the quality of the support, the monitoring suggests that the pillar was certainly adequately supported for the duration of the monitoring period. The data also indicates that the support is performing its reinforcement function well under the imposed stress change and deformation. Bolts installed in the pillar are still performing well because they are connected to a robust retention system (shotcrete and mesh) that prevents broken rock from ravelling.

Publications

NOTE: for a complete listing of the principal author's publications please go to: https://scholar.google.ca/citations?user=B-M3Ds4EAAAAJ&hl=en

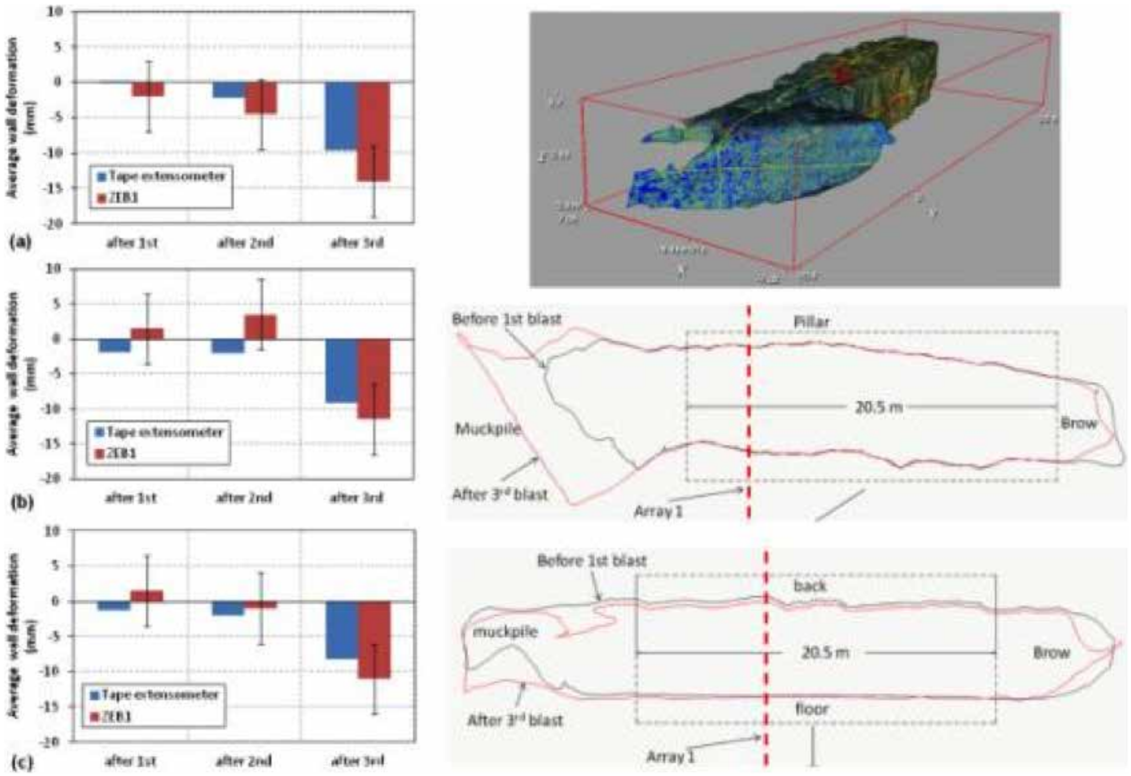
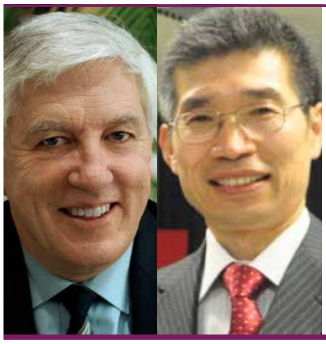


Figure 93: Validation of ZEB1 with tape extensometer measurements at three arrays (left) with Array 1 in top and determination of drift convergence adjacent to the monitored pillar (right) with point cloud (top), horizontal profile (middle), and vertical section (bottom)

5.6 Dynamic Support Research for Effective Ground Control

Team:
Laurentian University, MIRARCO – *Drs. Ming Cai* and *Peter K. Kaiser*



Titles: Dr. Peter Kaiser, Professor Emeritus; Dr. Ming Cai, Chair in Geomechanics, Laurentian University
Role: PI and Co-PI responsible for SUMIT Program technical delivery and project execution
Collaborators on team: Vale, Sudbury Integrated Nickel Operations – A Glencore Company, Rio Tinto

Abstract

This project aimed at consolidating research findings from various SUMIT sub-projects and of relevant research funded by other means including NSERC, CEMI and industrial sponsors (Sudbury Integrated Nickel Operations, a Glencore Company, LKAB, Newcrest, Rio Tinto and Vale). The product is an up-dated version of the 1996 Canadian Rockburst Support Handbook with special consideration given to excavation vulnerability and strainbursting (see Section 5.1).

Research outcome – Guide to Rockburst Support Selection

The Canadian Rockburst Program was completed in 1995 and the Canadian Rockburst Support Handbook published in 1996. This handbook presented a systematic engineering approach to select rock support for burst-prone mines by assessing both support demands and support capacities. After two decades, it was deemed necessary to update the handbook in the form of a "Guide to Rockburst Support Selection" and to enhance the support selection process by assisting practicing engineers in implementing systematic design procedures. This comprehensive guide should help ground control engineers to create safer work places in burst-prone mines.

In recent years, many new rock support components have been developed to enhance the ground control toolbox and new laboratory and field test data have become available to better assess the dynamic performance of rock support. Monitoring systems and data processing tools have been developed to better assess the seismic hazard and the risk of rockbursting and rockburst damage. Furthermore, it has been learned that strainbursts, whether self- or seismically triggered, often dominate the failure process of excavations when statically loaded by mining-induced stresses and seismically triggered by a remote seismic event. This strainburst and related damage process was not well reflected in the engineering approach presented in 1996. New insight into the role of strainbursts is now summarized in this Guide based on detailed discussions presented in various keynote lectures (e.g., Kaiser and Cai, 2013). Even though it is realized that further research will be required to arrive at yet more reliable means for energy release calculations to design support against seismically triggered strainbursts, this important aspect is included in a semi-qualitative manner in this revision.

Due to the complex behaviour of highly and dynamically stressed rock in hard rock mines as well as the complex interaction between rock support and stress-fractured rock, it is difficult to provide strict guidelines. Hence, this guide is not a "cook book" and the reader is expected to respect site-specific experiences when following the approach outlined in this guide. Nevertheless, it is a source of information that should assist in making prudent decisions.

Realizing that the interaction of geology, variability in rock mass properties, stress and dynamic factors leads to a highly complex process of rockburst damage and damage mitigation with ground support, a 3D support system needs an assessment and selection tool called BurstSupport, developed in collaboration with MIRARCO Mining Innovation. This tool is available from MIRARCO (www.mirarco.org).

Finally, both the 1996 version and this Guide rely for some design aspects on the use of scaling laws to anticipate ground motions from seismic sources. Mendecki (2016) presented the fundamentals for seismic hazard assessment and confirmed the approach as well as the parameters introduced in the 1996-version for PGV (peak ground velocity) prediction. He found that the parameters derived for a Canadian mine were applicable to other mines. When designing support, it is necessary to arrive at meaningful but not excessively conservative design inputs to predict extreme but realistic ground motion limits similar to 100-yr floods for dam design or 50-yr wind loads for high-rise design. The scaling law approach aims at providing such practically meaningful design inputs.

However, at many locations in a mine the actual ground velocities (PGVA) could be higher or smaller than the PGVD because of the effect of radiation patterns, wave reflections and attenuation. Because it is the actual PGV, the PGVA, that causes damage and not the extreme PGVD used for design, scaling laws with parameters intended for design are therefore not applicable for forensic analyses of rockburst damage. This important aspect is emphasized in this Guide.

When assessing rockburst damage situations, the effect of radiation patterns, wave reflections and attenuation must be respected and means to obtain the actual ground motions at the damage location must be adopted. For this purpose, a synthetic ground motion assessment tool, called S-GMAT, has been developed in collaboration with the Institute for Mine Seismology (IMS). It provides 3D radiation patterns from seismic point sources with measured source parameters from analytical or numerical solutions. S-GMAT is available for beta testing by program sponsors.

MIRARCO at Laurentian University is in the process of publishing a *"Rockburst Support Reference Book"* as a much enhanced update of the 1996 *Canadian Rockburst Support Handbook* (Kaiser et al., 1996).

Three volumes will be released in late 2017 and early 2018:

1. Rockburst phenomena and support characteristics
2. Rock support to mitigate rockburst damage caused by dynamic excavation failures
3. Rock support to mitigate rockburst damage caused or dominated by dynamic disturbances from remote seismicity

Preliminary manuscripts will be distributed to interested parties offering timely feedback and contributions for possible inclusion.

Final copies will be distributed free of charge upon registration as a reader. ISBN numbers are available from the authors. Further, the guide introduces two tools (BurstSupport and S-GMAT) for support selection and rockburst damage assessment.

References Cited

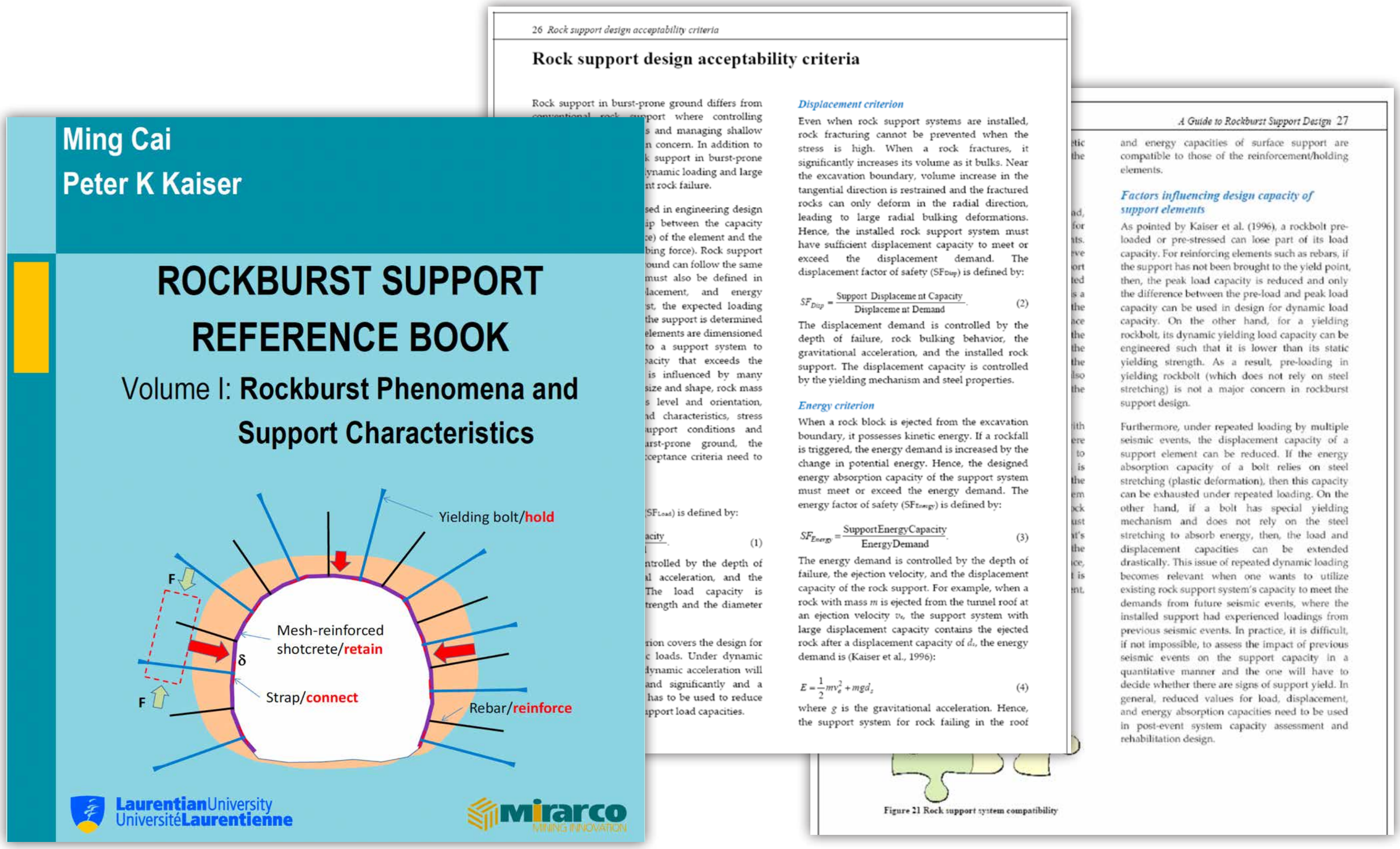
Mendecki, A.J. (2016). Mine Seismology Reference Book – Seismic Hazard. Institute for Mine Seismology, free from http://www.imseismology.org/wp-content/uploads/2015/06/SH-cover.png, 88p.

Kaiser, P.K. & M. Cai (2013) Critical review of design principles for rock support in burst-prone ground – time to rethink! Keynote Lecture. In: Ground Support 2013, Potvin, Y. and Brady, B. (editors), p.3-38.

Selected Publications

Cai, M. and P.K. Kaiser (2017). Guide to Rockburst Support Selection. To be published by MIRARCO in mid 2017.

NOTE: for a complete listing of the authors' publications please go to:
https://scholar.google.ca/citations?user=BM3Ds4EAAAAJ&hl=en for kaiser and
https://scholar.google.ca/citations?user=kHg_k5kAAAAJ&hl=en for Cai



Ming Cai
Peter K Kaiser

ROCKBURST SUPPORT
REFERENCE BOOK
Volume I: **Rockburst Phenomena and Support Characteristics**

26 *Rock support design acceptability criteria*

Rock support design acceptability criteria

Displacement criterion

Even when rock support systems are installed, rock fracturing cannot be prevented when the stress is high. When a rock fractures, it significantly increases its volume as it bulks. Near the excavation boundary, volume increase in the tangential direction is restrained and the fractured rocks can only deform in the radial direction, leading to large radial bulking deformations. Hence, the installed rock support system must have sufficient displacement capacity to meet or exceed the displacement demand. The displacement factor of safety (SF_{Disp}) is defined by:

$$SF_{Disp} = \frac{\text{Support Displaceme nt Capacity}}{\text{Displaceme nt Demand}} \quad (2)$$

The displacement demand is controlled by the depth of failure, rock bulking behavior, the gravitational acceleration, and the installed rock support. The displacement capacity is controlled by the yielding mechanism and steel properties.

Energy criterion

When a rock block is ejected from the excavation boundary, it possesses kinetic energy. If a rockfall is triggered, the energy demand is increased by the change in potential energy. Hence, the designed energy absorption capacity of the support system must meet or exceed the energy demand. The energy factor of safety (SF_{Energy}) is defined by:

$$SF_{Energy} = \frac{\text{SupportEnergyCapacity}}{\text{EnergyDemand}} \quad (3)$$

The energy demand is controlled by the depth of failure, the ejection velocity, and the displacement capacity of the rock support. For example, when a rock with mass m is ejected from the tunnel roof at an ejection velocity v_e, the support system with large displacement capacity contains the ejected rock after a displacement capacity of d_s, the energy demand is (Kaiser et al., 1996):

$$E = \frac{1}{2}mv_e^2 + mgd_s \quad (4)$$

where g is the gravitational acceleration. Hence, the support system for rock failing in the roof

Figure 21 Rock support system compatibility

A Guide to Rockburst Support Design 27

and energy capacities of surface support are compatible to those of the reinforcement/holding elements.

Factors influencing design capacity of support elements

As pointed by Kaiser et al. (1996), a rockbolt preloaded or pre-stressed can lose part of its load capacity. For reinforcing elements such as rebars, if the support has not been brought to the yield point, then, the peak load capacity is reduced and only the difference between the pre-load and peak load capacity can be used in design for dynamic load capacity. On the other hand, for a yielding rockbolt, its dynamic yielding load capacity can be engineered such that it is lower than its static yielding strength. As a result, pre-loading in yielding rockbolt (which does not rely on steel stretching) is not a major concern in rockburst support design.

Furthermore, under repeated loading by multiple seismic events, the displacement capacity of a support element can be reduced. If the energy absorption capacity of a bolt relies on steel stretching (plastic deformation), then this capacity can be exhausted under repeated loading. On the other hand, if a bolt has special yielding mechanism and does not rely on the steel stretching to absorb energy, then, the load and displacement capacities can be extended drastically. This issue of repeated dynamic loading becomes relevant when one wants to utilize existing rock support system's capacity to meet the demands from future seismic events, where the installed support had experienced loadings from previous seismic events. In practice, it is difficult, if not impossible, to assess the impact of previous seismic events on the support capacity in a quantitative manner and the one will have to decide whether there are signs of support yield. In general, reduced values for load, displacement, and energy absorption capacities need to be used in post-event system capacity assessment and rehabilitation design.

Front cover example page format for the Guide to Rockburst Support Selection updated handbook

SUMIT #6

Sustainable Operations – Energy Management and Ventilation

6.1
Seasonal Thermal Generators

6.2
Optimal Mine Site Energy Suppky (OMSES)

6.3
Mine to Bullion Energy Audits

6.4
Integrated Technologies

Sustainable Operations – Energy Management and Ventilation

Team:
Laurentian University – *Dr. Dean L. Millar* with *Sidney Schafrik, Alberto Romero, Michelle Levesque, Harvard Farrant* and:
Olimpia Banete, Donato Grandal-Vilar, Wendy Matthews, Ian Berdusco, Curtis Cameron, Onneile Thomas, Cyle Wheeldan,
Robert Malette, Johannes Adam, Ana Leite, Felipe Gabriel, Adam Turcotte, Tim Doan, Valeria Pavese, Julia Andrade, Eden Laurindo,
Diogo Sanchez de Oliveria, Angelina Rocha, Flavia Vespucio, Justin Bontin, Jordan Gladu, Alex Hutchinson, Gabriel Janakavaj
with *Post Doc., Dr. Monica Carvalho*

Title: Dr. Dean Millar
Role: Co-PI
Collaborators on team: Vale, Sudbury Integrated Nickel Operations – A Glencore Company, MTI[6], BBE

Motivation

Energy costs as a proportion of overall production costs for mines has been steadily increasing from about 5% to 16% from 1960 to 2010 (Figure 94). At 18%, this proportion is higher for non-metal mines.

Abstract

The formal name of the sub-project for this work was: Sustainable Operations – Energy Management and Ventilation-on-Demand. At the outset, this itself was divided into broad categories of work: i) development of Ventilation-on-Demand, ii) Demand side energy conservation, iii) utilization of waste heat recovery and storage and iv) innovation in on-site supply side management.

The research team made huge strides in improving modelling capability and understanding of seasonal thermal regenerators, which reduce the amount of natural gas that must be paid for heating mine air in the winter and avoids the installation of mechanical refrigeration capacity with its concomitant electricity consumption and demand for capacity on the electrical grid. The annual savings for operators arising from seasonal thermal regenerators were shown to run to millions of dollars.

Even higher economic value to operators was identified through the Optimal Mine Site Energy Supply design toolkit developed as a part of the work. This incorporated developments in mine energy auditing and the visualization of the results of such auditing tasks with 4D Sankey diagrams. The audits undertaken as part of the work identified and visualized energy savings of $140 million per annum (2008 U.S) for a pyrometallurgical operation and $eight figures per annum for a processing/beneficiation operation. It was harder to undertake energy audits for underground mines, but methodologies were developed to do this in a standardized fashion.

Importantly, for some mine operations where sub metering is not available, the work developed a smart underground mine electricity monitoring system that drew upon artificial neural network technology to develop the end-use consumption breakdown from these single meter readings: the ‘smarts’ were in the neural network. A detailed auxiliary ventilation duct computer simulation tool was created that explicitly accounted for the leakiness of these systems, practically observable in the field. The tool demonstrated that leakage can have a dramatic effect on ventilation-on-demand economics and that this absolutely needs to be taken into account in auxiliary ventilation system design. Varying fan speed (as in VoD) is one way to save energy and money in auxiliary ventilation systems, another way is to design a regulator device that, as well as permitting adjustment of the flow, can recover the energy in the air flow that would otherwise be dissipated. This is the subject of the last topic of research reported.

As will be appreciated from the above, much, if not all of the undertaken in the SUMIT 6 package relied on techno-economic analysis tools that required economic input, principally to guide observation taking, analysis and design tasks. Thus, in the report that follows, what was discovered about mine energy economics because of the research, is reported first.

Context

Much of the work undertaken in the SUMIT 6 package relied on techno-economic analysis tools that required economic input. In trying to establish the portrait of energy economics as applicable to the mining industry, it quickly became evident that there were insufficient, or highly fragmented, public domain data for informed decision making for the Canadian sector as a whole. Consequently, one early task the team undertook was to consolidate and defragment multiple data sources from Canadian federal and provincial government departments, so that a coherent energy economic picture for the country's mining sector could be established. The main findings were:

In real terms (Figure 94),
- electricity costs have remained relatively steady since 1990
- natural gas costs peaked around 2004 and have declined since, driven by fracked production
- the cost of fuel oil, on which the competitiveness of the mining industry so strongly depends, had significantly increased.

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[6] *Now Joy Global*

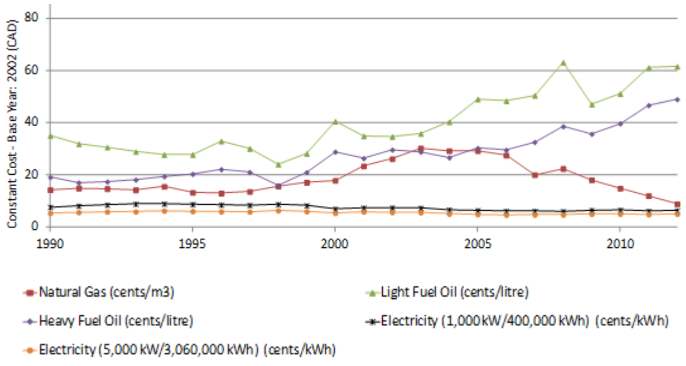


Figure 94: Trends in energy pricing for the Canadian mining industry 1990 to 2012

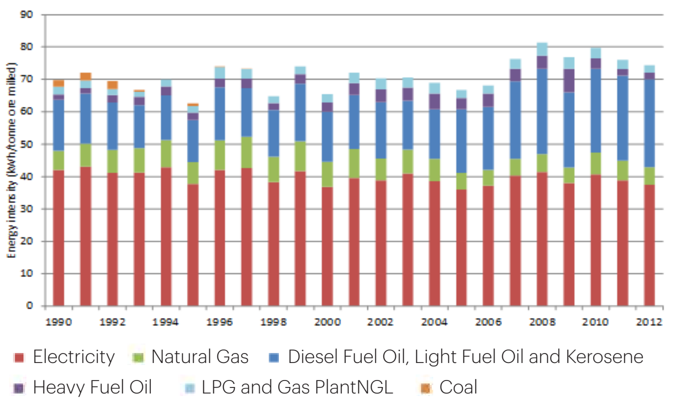


Figure 95: Relative proportions of energy consumed per tonne of ore mined for the Canadian base metals mining sector 1990 to 2012

There is an appreciable lag between the time at which energy data is collected by Government departments and the time at which is has been verified and released to the public domain.

Across the base metals sector, the average consumption of electricity per tonne of ore produced has remained the same or declined very slightly since 1990 (Figure 95). This is surprising because over the same period, there has been greater usage of diesel powered mobile equipment in the sub-surface, and one would expect electricity consumption to increase due to higher ventilation requirements. The fact that it has not increased could be due to electricity conservation efforts in the mine working well, with the electricity consumption savings 'being taken' in the form of the enhanced production attainable with the diesel equipment by mining industry operators.

Energy costs' proportion of overall production costs for mines has been steadily increasing from about 5% to 16% from 1960 to 2010 (Figure 96). At 18%, this proportion is higher for non-metal mines.

Since the 1970s, the types of energy conservation measures applied at operating mines have remained more or less the same to the present day, with the exceptions of i) increased metering and process control (smart measurements and integration of technologies) and ii) adoption of renewable energy technologies, when appropriate.

Selected References:

Millar, D.L. and Hassani, F., 2015. Editorial. Special Issue on Energy in Mining. Applied Thermal Engineering. Vol 90, p.1090-1091. doi: 10.1016/j.applthermaleng.2015.07.053.

Levesque, M., Millar, D.L., and Paraszczak, J., 2013. Energy and Mining – The Home Truths. Journal of Cleaner Production. Vol. 84, p.233-255.

Millar, D.L., Levesque, M., Lyle, G., and Bullock, K., 2012. Enabling progressive energy management practice for minerals operations. CIM Journal, Vol 3. No. 3, p.178-184.

Work on the SUMIT 6 project was undertaken by executing a suite of inter-related sub-projects:

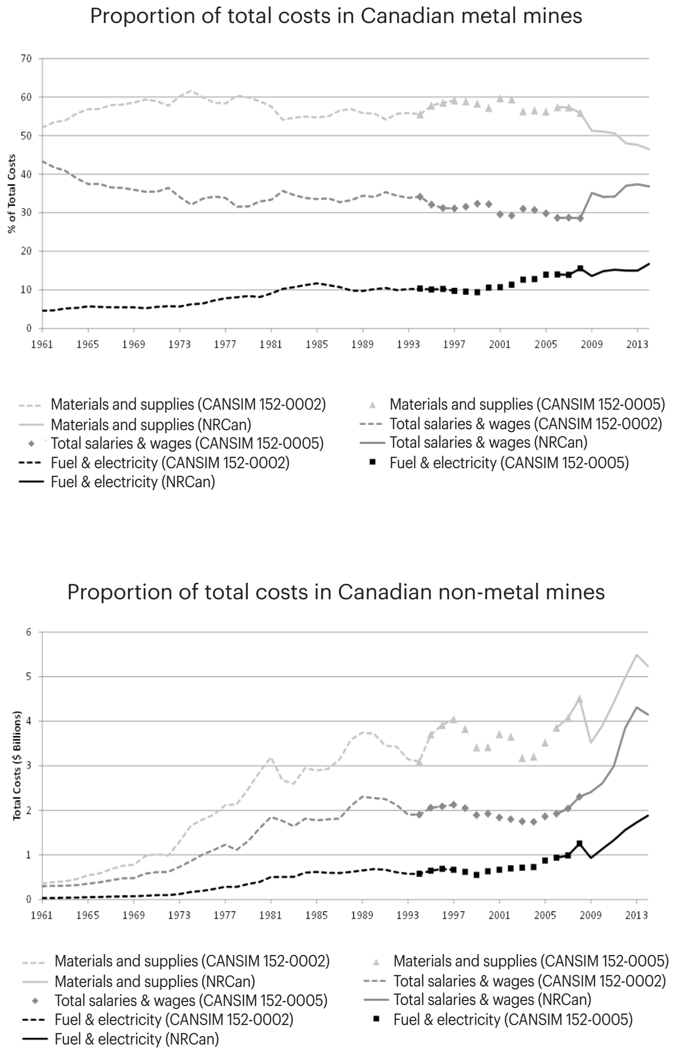


Figure 96: Relative proportions of total cost of production for the Canadian metals and non-metals mining sector 1961 to 2010 (various sources)

6.1 Seasonal Thermal Generators

At Vale's Creighton Mine in Lively, Ontario, it has been common practice for decades to draw all (~800 m3/s) of the mine's ventilation air through a one km3 scale body of fragmented rock (Figure 97). During the summer, the heat in the air is stored by the broken rock fragments as the former is cooled for delivery to the sub-surface. In the winter, the heat stored in the rocks during the summer warms the cold winter air as it passes through the same fragmented rock body.

Methodology

The work done on this topic concentrated on computational fluid dynamics simulation of the system to enhance understanding of its behaviour to improve the value of the asset as a ventilation air cooling and heating system at Creighton specifically and to permit design of new systems, that we have called seasonal thermal regenerators, at other mine sites where opportunities allow. From an energy point of view, the key advantage of systems such as these is that they displace active heating or refrigeration systems that would otherwise have to be installed and operated at great cost.

Summary of Findings

The simulation work revealed that the equivalent active heating or cooling capacity of what is, fundamentally, a pile of broken rock, is approximately 20 MWth. Simulations also led to proposals for engineering measures that would extend this capacity.

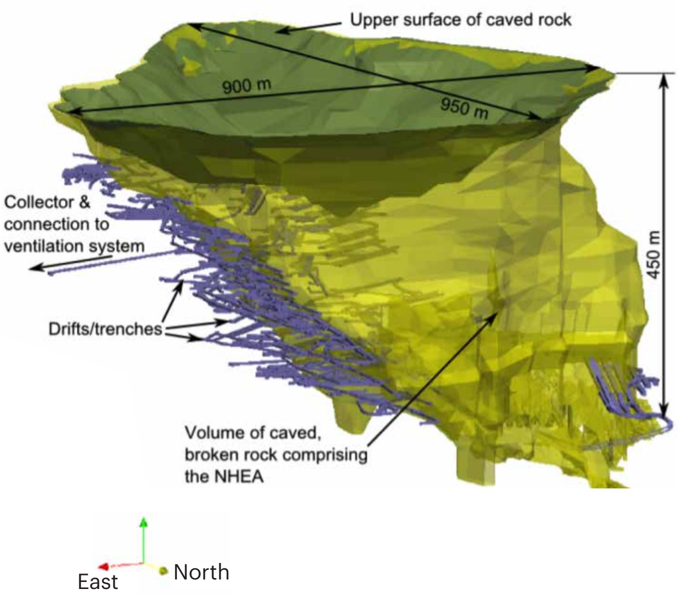
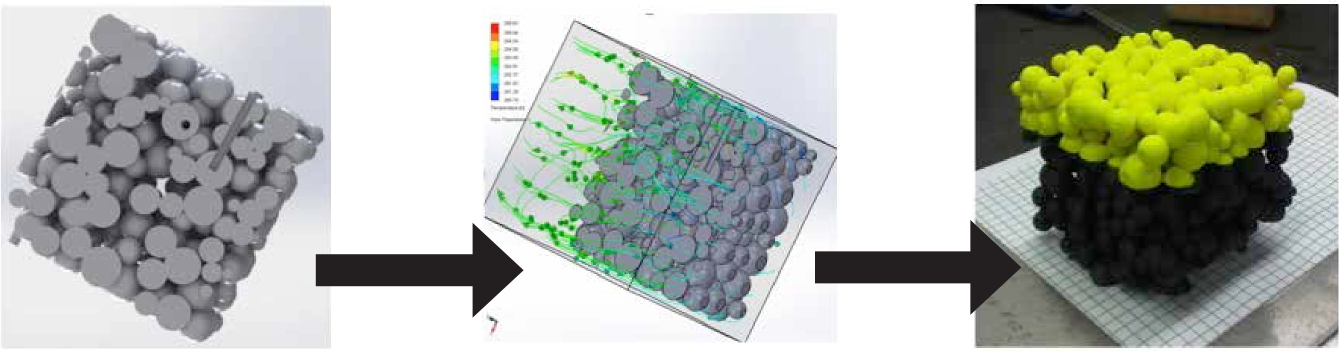


Figure 97: 3D rendering of Creighton Mine's so-called, Natural Heat Exchange Area (NHEA), an example of a large scale seasonal thermal regenerator

Team Contributions

PhD. Natural Resources Engineering, Sidney Schafrik (Dec 2014, Geometry of heat exchange in fractured & broken rocks). His PhD work concentrated on establishing the thermal and flow parameters of the body of fragmented rock, and he did this by development of an entirely novel methodology that integrated CAD/CAM (including rapid prototyping) and CFD techniques and verified this with site observations (Figure 98).

M.A.Sc. Natural Resources Engineering, Olimpia Banete (Apr 2014, Towards modeling heat transfer using Lattice Boltzman Machine for porous media). Olimpia set up, and established LBM models of porous media within MATLAB. She then undertook laboratory experiments of heat transfer in gravel scale materials in order to verify her numerical results, and these are published in her Masters thesis.



CAD/CAM

CFD

Physical Model & Wind

Figure 98: Illustration of the CAD/CAM/CFD integrated methodology that led to identification of air flow and heat transfer parameters for Vale's NHEA, undertaken by Sidney Schafrik

Donato Grandal-Villar (CFD Modelling of the Natural Heat Exchange Area). Donato creatively explored the use of Computational Fluid Dynamics of broken rock piles as methods of heat and coolth storage for architectural structures such as sports stadiums, swimming pools and other similar civic facilities (Donato being a Professional Architect and Urban Planner). Donato's work was seminal in visualizing the dynamic behaviour of seasonal thermal regenerative heat stores. It was through Donato's work on this that it was discovered that the NHEA exhibits a seasonally driven thermal wave like behaviour (apparent in the recurring seasonal spatial distributions of rock temperature within the NHEA, Figure: 99).

It was Dean who realized that the phenomenon could be engineered: Figure 99, on the right-hand side, shows how the behaviour is modified through the inclusion of a cap to lengthen the flow path of the air through the NHEA. 'Pulses' of extreme hot and cold portions of rock are apparent on the RHS of the NHEA, but on the LHS, the warm and cold rock temperature fronts (blue and red) have 'spread out', such that at the doors on the LHS of the NHEA, the only temperature (green) 'seen' at this location is the average temperature, year round. This is exactly the desired effect, currently only approximated through control door manipulation, with the added benefit of enhanced volumes of air flow because all the control doors are opened and offer less resistance to the fans. The mine operator needs to have greater confidence and understanding of this behaviour before committing to engineering measures involving a cap, and the simulations have helped in this regard.

M.Eng. Renewable Energy, Wendy Matthews (Jun 2015, CFD modeling of Creighton Natural Heat Exchange Area). Established the detailed methodology for calibrating the heat transfer and air flow properties of the NHEA within a mainstream Computational Fluid Dynamics (CFD) code.

Bob Anderson. Software implementation of a procedure to back-calculate the temperatures of the rock in which the NHEA heat is stored, from field observations of air temperatures at the peripheral control doors of the NHEA.

Thus, a concept of thermal tomographic inversion was born. Bob managed to execute the concept completely in 2D, based on a line integral formulation for the unknown rock temperatures (Figure 100). Ultimately it was found that the proposed technique is not that robust due to the great thermal inertia of the rock ma-

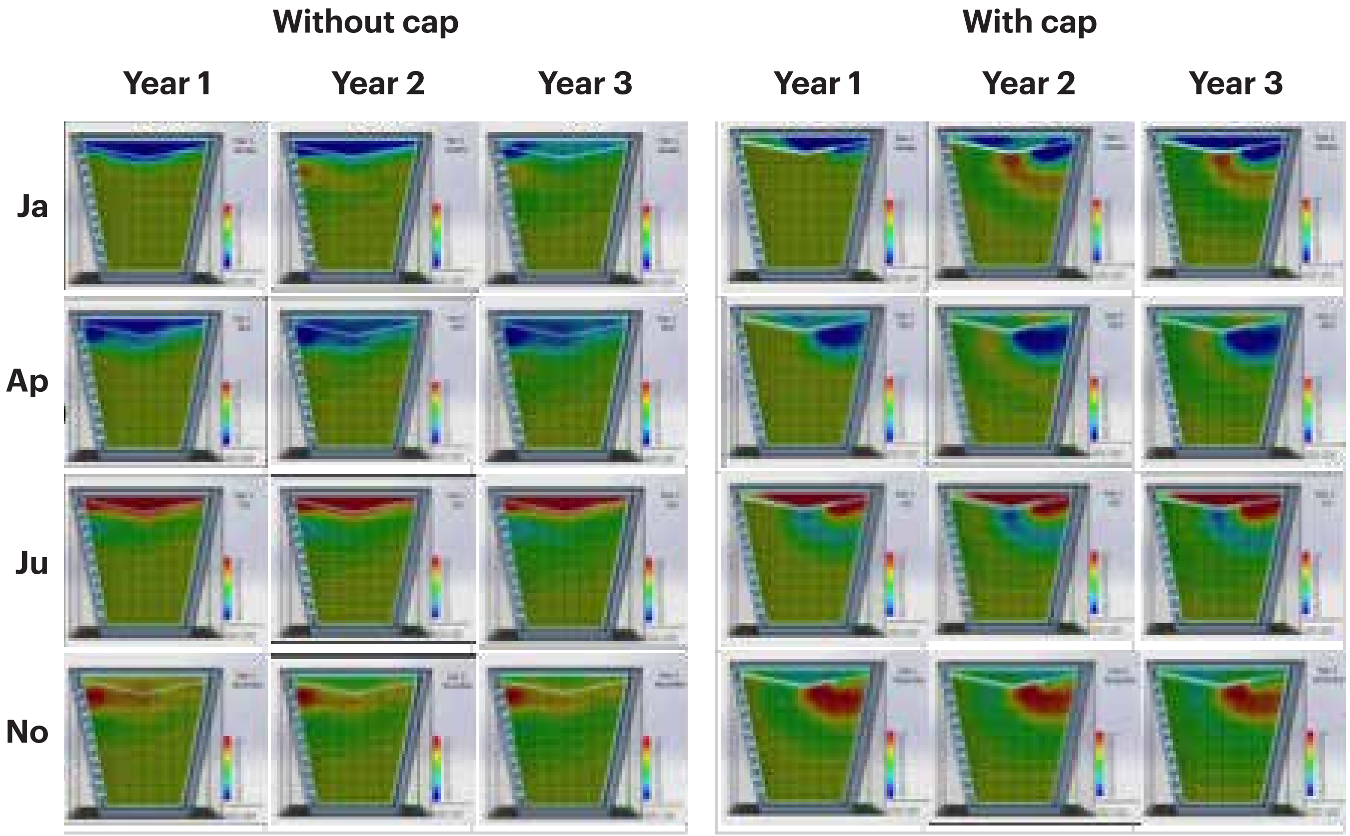


Figure: 99: Seasonal snapshots of the temperature field within a portion of fragmented rock 1/10th of the volume of Creighton Mine's NHEA. Air flow is from the top, horizontal surface to the steeply dipping LHS surface containing gaps denoting air flow control doors. LHS set (of 12) showing the seasonal temperature patterns when air flow can cross the whole of the top horizontal surface. RHS set (of 12) showing the seasonal temperature patterns when air flow can cross only the RHS of the top horizontal surface

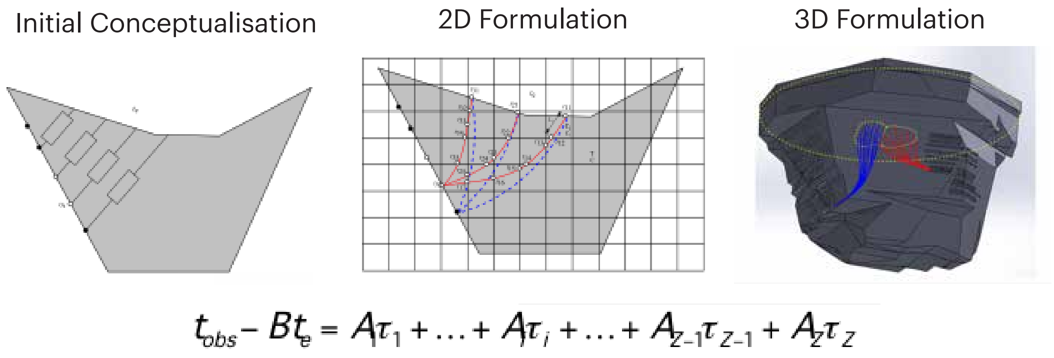


Figure 100: Refinement progression of heat transfer conceptual models for Creighton Mine's NHEA. Left: 'Slusher trench' model. Middle: 2D line integral heat exchange model. Right: 3D line integral heat exchange model

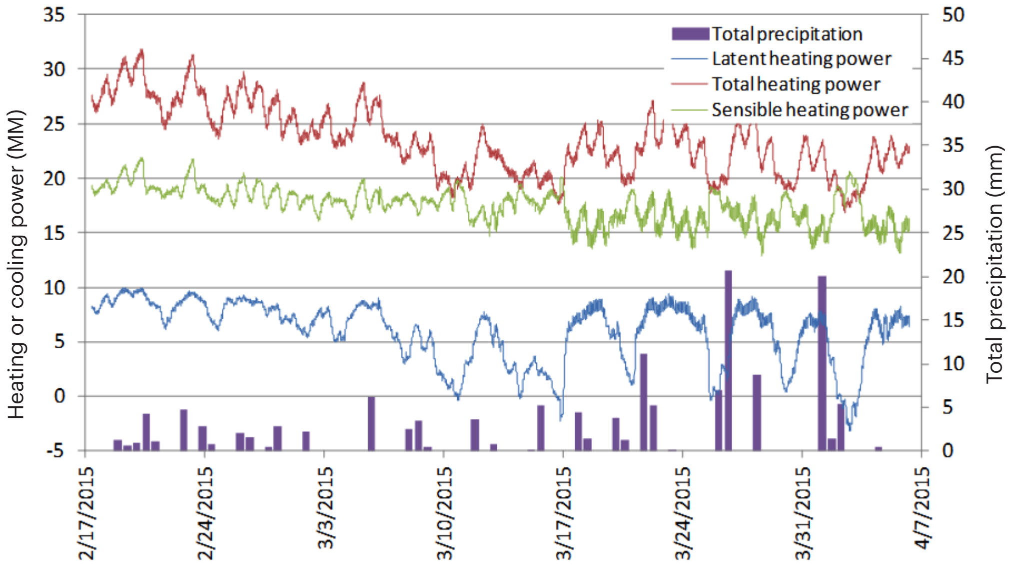


Figure 101: Integration and reduction of data from logged temperature and humidity sensors from the 800 level at Vale's Creighton Mine and data from Environment Canada to produce time series of latent, sensible and total heating or cooling

terial and the lack of it for the air. 3D implementation, further air flow control door optimization, and investigation of latent heating and cooling in the NHEA is underway supported by a follow-on UDMN project.

Ms Cheryl Allen, Vale. Cheryl Allen was MIRARCO's single point of contact on research concerning the Natural Heat Exchange Area at Creighton Mine. Cheryl and her team at Creighton (Doug O'Connor, Brian Keen and Paul Aho. have provided abundant data on temperatures, pressures, humidity and air flow rates through the NHEA through access to their PI system, shown in Figure 101.

We believe that this access was completely instrumental to establishing new scientific understanding of how seasonal thermal regenerators operate at ~one km^3 scale.

Selected References

Schafrik, S. & Millar, D.L., 2015. Verification of a CFD code use for air flow simulations of fractured and broken rock. Applied Thermal Engineering, Vol. 90, p.1131-1143. doi:10.1016/j.applthermaleng.2015.03.021.

6.2 Optimal Mine Site Energy Supply (OMSES)

This part of the SUMIT 6 work focussed specifically on optimizing the integration of conventional and novel energy technology, motivated by the imperative of minimising energy costs for mine operators.

Methodology

The tools developed initially adopted mixed integer linear programming techniques to minimise lifecycle (discounted) cost of the supply of energy to a mine site defined by demand time series for energy in all its forms (electricity, gas, diesel, steam, hot water, chilled water, etc.). The mathematical programming formalism introduced constraints reflecting mass and energy balances across individual technologies (e.g. the heat provided to a diesel engine in the form of fuel must equal the shaft work output plus the heat added to cooling water and/or cooling oil plus the heat added to the air around the engine – exactly), and across each energy utility provided (e.g. the total electricity produced by all generating devices plus imported electricity must equal the electricity demand from the mine operations). Graphically such considerations are represented in Figure 102.

Summary of Findings

Ultimately the tool was proven to be able to integrate renewable technologies exploiting solar, hydro, wind and biomass energy alongside conventional prime movers such as gas turbines, steam turbines and diesel engines. It was also extended to incorporate energy storage systems (including: batteries, pumped storage, biomass bunkers) into the optimal energy supply solution.

Finally, Model Predictive Control algorithms were integrated into the optimal energy supply solution procedure so that the robustness of this solution in the face of real world uncertainties of energy demand (e.g. meteorological) and the economic environment could be improved and maintained. Overall, a portfolio of tools has been developed that together exceed the capabilities of many mainstream software tools for energy supply system design.

Each time the tools have been applied to real world mining

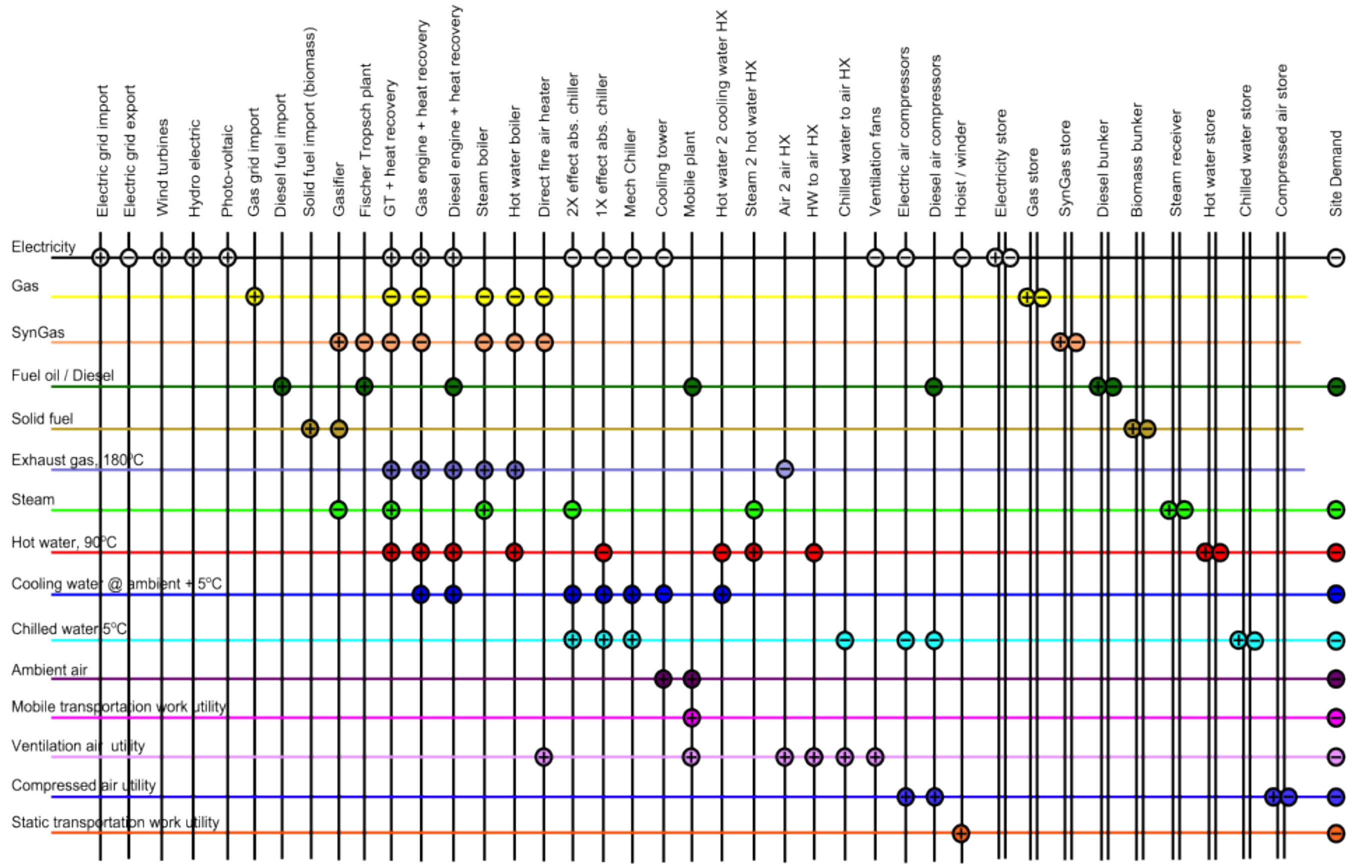


Figure 102: Conservative superstructural grid for an Optimal Mine Site Energy Supply problem involving 15 distinct types of energy supplied to a mine and 30 conventional and renewable energy conversion technologies with nine distinct types of energy storage technologies. Energy produced by a technology is denoted by a plus '+'. Energy consumed by a technology is denoted by a minus '-'. Consumption nodes on the extreme RHS denote the mine demand

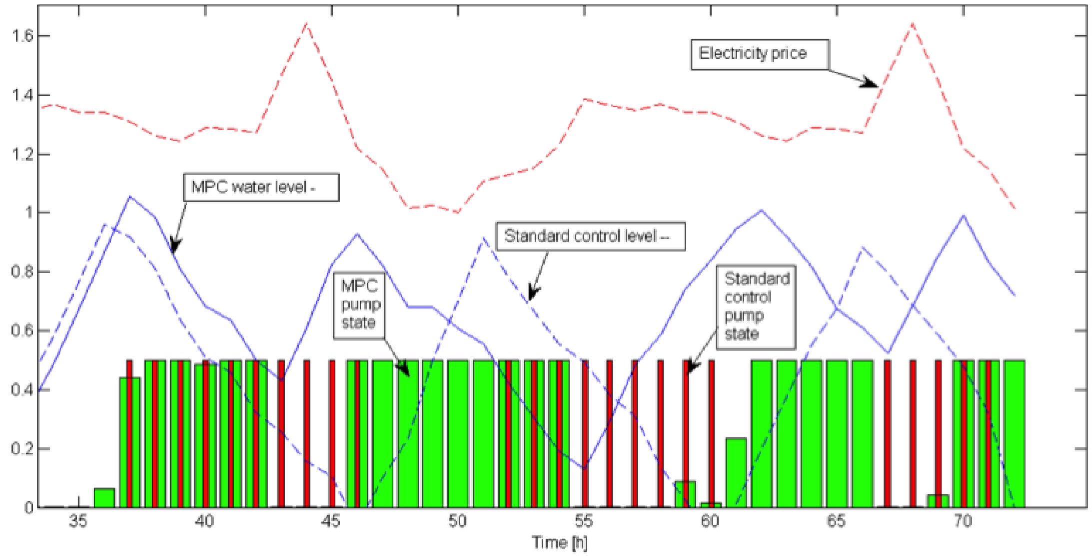


Figure 103: Results of simulation of an OMSES solution for optimal dewatering of an underground mining operation using Model Predictive Control (MPC). The dewatering system uses the mine sump capacity as an energy store so that the pumps avoid operating at times when the price of electricity is highest

case studies, they have returned reductions in life cycle energy costs for mines of between 15% and 50% compared with base case BAU scenarios, with the greater amount applying to remote mines and communities who choose to consolidate their demand with mines. The tool has also been used to i) establish designs for mines operating with 100% renewable energy and ii) answer key design questions for mines such as what rating should their electrical connection be or how big should their gas pipe be – for minimum cost.

Team Contributions

Post Doctoral Fellow: Dr. Monica Carvalho. During her tenure at MIRARCO, Dean and Monica applied for, and were successful in securing, a NSERC IRDF (Industrial Research and Development Fellowship) to co-fund Monica's position. Monica represented the team at several international conferences, and, as part of the collaboration with Dr. Andy Fyon, of the Ontario Geological Survey, flew to Fort Severn First Nation in the Far North of Ontario to gather data for demand consolidation energy supply modeling involving mine and First Nations communities.

PhD. Natural Resources Engineering, Alberto Romero (Aug 2016, Optimal Mine Site Energy Supply). Alberto has extended the Polygeneration Optimization methodology and integrated the ideas of model predictive control with those of OMSES to ensure mine site energy supply remains optimal in the face of environmental and energy market uncertainties. An example of optimal pump scheduling results in the face of uncertainty in precipitation and market electricity price is presented in Figure 103.

B.Eng. Mining Engineering. Research Project Students (2012)

These students undertook some preliminary conceptual feasibility optimization work in the area of mine dewatering.

– Ian Berdusco (MPC Algorithm for mine dewatering)
– Curtis Cameron (MPC Algorithm for mine dewatering)
– Onneile Thomas (MPC Algorithm for mine dewatering)
– Cyle Wheeldan (MPC Algorithm for mine dewatering)

Mr. Marc Boudreau and Mr. Dan Dumais of BESTECH provided a industrial secondment opportunity for Alberto Romero. This provided Alberto with exposure to VOD control and measurement equipment (such as presented in Figure 104) typical demands, and demand profiles, from outside the Sudbury Basin, that have been instrumental to his PhD research on Optimal Mine Site Energy Supply, and the application of Model Predictive Control to polygeneration optimization.

Figure 104: BESTECH's NRG1-Eco system to minimize electricity consumption costs associated with auxiliary mine ventilation systems.
Source:http://www.bestech.com/Downloads/ProductSheets/BESTECH_NRG1-ECO.pdf

Mr George Shields, De Beers were not identified as a formal mining industry collaborative partner at the time of application. Nevertheless, interactions with George Shields, and De Beers Victor Mine in the Far North of Ontario, led to deep insights in to the specific needs of remote mines in Northern Ontario, and how these can be met at the same time as meeting the needs of other remote communities in the same region. Interactions with George, led to a joint publication on OMSES.

Dr. Andy Fyon, Ontario Geological Survey. Dr. Fyon provided an opportunity for Dr. Monica Carvalho to visit the Fort Severn First Nation community to share experience and practice of optimal energy supply, including renewables, for remote communities. Monica's demonstration of thin film floating photovoltaic systems remains memorable for all members of this community.

Selected References

Romero, A., Carvalho, M. and Millar, D.L., 2016. Optimal design and control of wind-diesel hybrid energy systems for remote arctic mines. ASME Journal of Energy Resources Technology, doi:10.1115/1.4033677.

Romero, A., Chacartegui, R., Becerra, J.A., Carvalho, M., & Millar, D.L., 2015. Analysis of the start-up and variable load operation of a combined cycle power plant for off-grid mines. International Journal of Global Warming. (In press).

Romero, A., Millar, D., Carvalho, M., Maestre, M., Camacho, E. F., 2015. A comparison of the economic benefits of centralized and distributed model predictive control strategies for optimal and sub-optimal mine dewatering strategies. Applied Thermal Engineering. Vol. 90, p.1172-1183, doi:10.1016/j.applthermaleng.2015.01.031.

Millar, D.L., Romero, A., Carvalho, M., Levesque, M., 2014. Optimal Mine Site Energy Supply. Book Chapter in Responsible Mining, Case Studies in Managing Social & Environmental Risks in the Developed World, Jarvie-Eggart, M.E., (ed), Society of Mining Engineers. ISBN 978-0-87335-373-1.

Carvalho, M., Romero, A., Shields, G. and Millar, D.L., 2014. Optimal synthesis of energy supply systems for remote open pit mines. Applied Thermal Engineering. Vol. 64, p.315-330.

Romero, A., Carvalho, M., Millar, D., 2013. Application of a polygeneration technique for a hospital in Northern Ontario. Transactions of the Canadian Society for Mechanical Engineering. Vol. 38, No. 1.

Carvalho, M., and Millar, D.L., 2012. Concept development of Optimal Mine Site Energy Supply. Energies. Vol. 5 No. 11, p.4726-4745.

6.3 Mine to Bullion Energy Audits

"You can't manage what you can't measure", so goes the famous saying coined by leaders in management science. In the case of energy management for the mines with whom we worked in detail on SUMIT 6 work, we realized that some mines had very sophisticated systems for measuring energy consumption that far exceeded the a priori expected capabilities.

Although there were some mining operations with whom we worked that effectively only had one electricity meter at the mine gate, other mines (who were very sensitive to the economic incentives available for mine operators in Ontario) had installed sensors in a very comprehensive manner throughout their operations in order to save themselves money via their tariffs. However, for these mines, other than seeing their billing diminish, on occasion opportunities for more detailed analysis of the data to save mine operators even more money were being missed; there was potentially a lack of interpretation rather than a lack of data.

Methodology

Using data from a main mine electricity meter, together with detailed knowledge of the breakdown of electricity consumption at the same facility, a method was developed that integrates signal processing technology and artificial neural network technology to identify, from the main mine meter data only, the times when the main mine hoist operated and the amount of electricity consumed by the hoist. The technique relied on applying a Fast Fourier Transform to the electricity meter data so that the spectral signature of hoist operation could be disaggregated from the meter data (Figure 105). This work is completely novel.

Summary of Findings

The work conducted on this topic can be summarized by saying that for both types of mines (metered and un-metered), SUMIT 6 integrated technologies to produce tools that improved the interpretation of energy data (Figure 106) that was being gathered, however extensive or narrow this was.

Team Contributions

M.A.Sc., Natural Resources Engineering, Michelle Levesque (Dec 2011, Understanding Mine Heat, Sankey Diagrams). At the outset of the project, Michelle worked on her Master's thesis that focussed on energy management of pyrometallurgical processes used in the reduction of nickel laterite ores to nickel bullion. This work identified energy savings of US2008 $140 million per annum for the asset operator (Figure 107).

PhD. Natural Resources Engineering: Michelle Levesque (submitted May 2015, Improved Energy Management for Mining). Michelle investigated the potential for electricity savings through

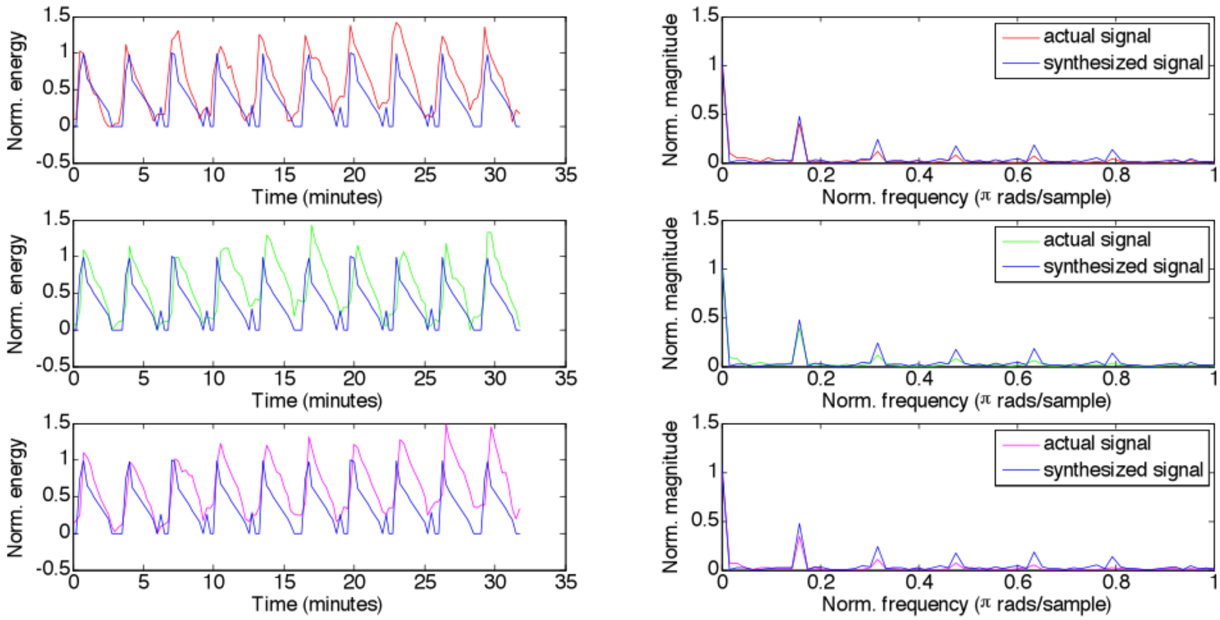


Figure 105: Identifying the 'fingerprint' of mine hoist operation within main mine electricity meter data, so that the hoist usage may be audited as part of a mine energy management plan. This technique is useful when only a single meter is installed at a mine site (which is frequently the case), and the usage of a particular centre of operations needs to be disaggregated for energy management

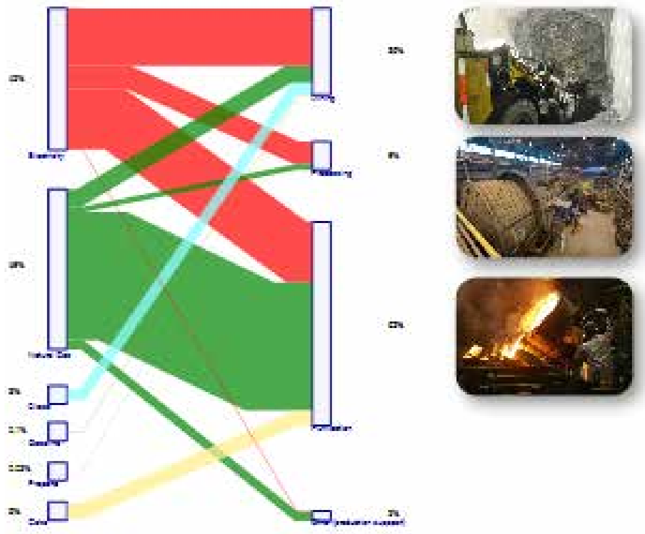


Figure 106: A Sankey diagram for energy consumed in support of nickel production from underground mines. The thickness of the ribbon reflects the rate of energy being consumed

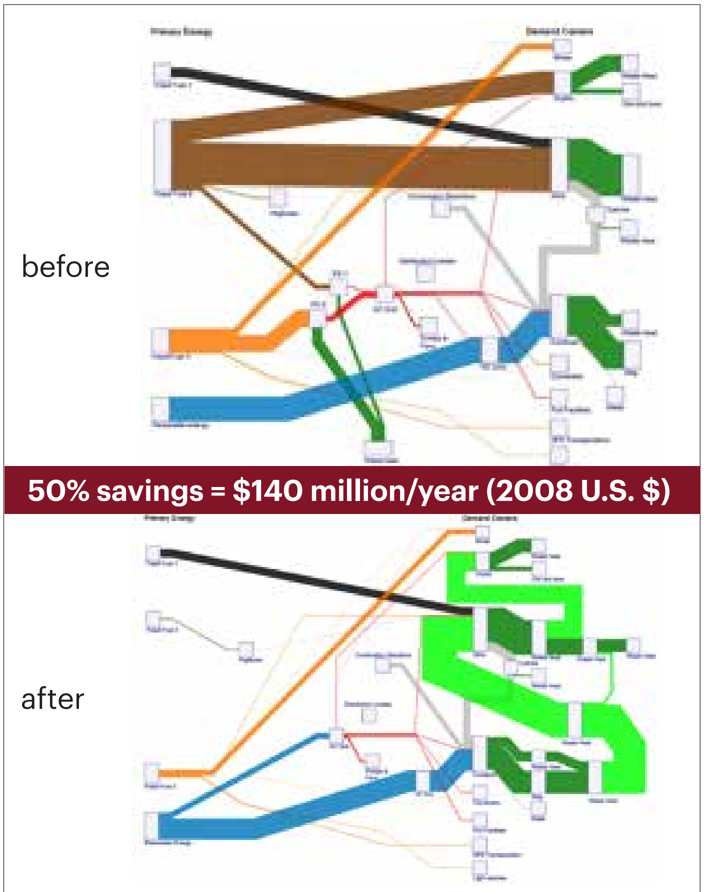


Figure 107: A Sankey diagram to illustrate the energy savings achievable for one plant operator through a more integrated approach to mine energy management

the use of plastic ducting for auxiliary mine ventilation, incorporating Ventilation-On-Demand approaches (see section 6.4). This work evolved to specification of a new method for simultaneous measurement of the friction factor and leakage factor for auxiliary ducting and fan systems.

The "run when full" work completed as part of Michelle's work also demonstrated the benefits of running beneficiation plants at their design rating (Figure 108).

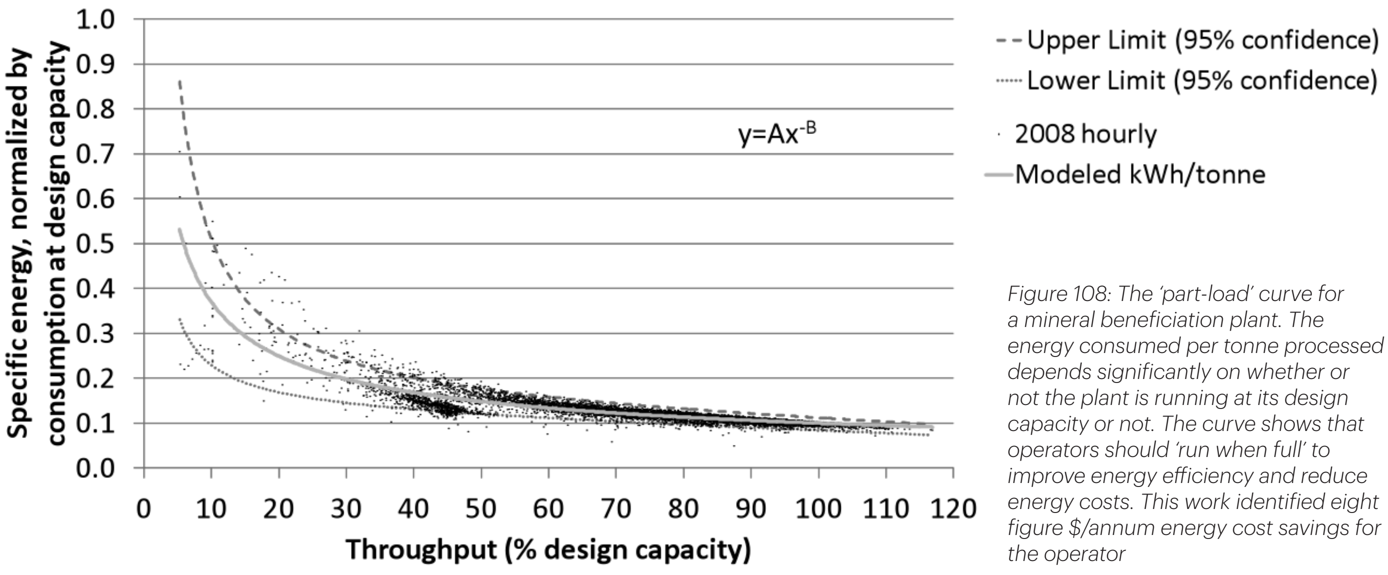


Figure 108: The 'part-load' curve for a mineral beneficiation plant. The energy consumed per tonne processed depends significantly on whether or not the plant is running at its design capacity or not. The curve shows that operators should 'run when full' to improve energy efficiency and reduce energy costs. This work identified eight figure $/annum energy cost savings for the operator

M.Eng. Natural Resources Engineering, Robert Mallette (May 2014, Garson Mine Energy Audit). Undertook an all fuels energy audit of Garson Mine, and work has proven very valuable as a resource for the other members of the group. Michelle made excellent use of the opportunity to provide guidance to Robert, based on her research, on the right methodology to use to undertake an all fuels energy audit of a modern mine (Figure 109).

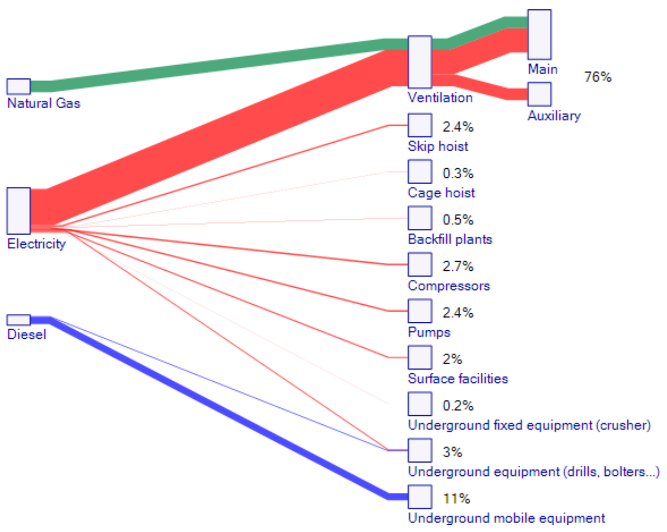


Figure 109: Sankey diagram for an underground mining operation producing a base metal product

M.Eng. Renewable Energy, Johannes Adam (ongoing, Sankey Diagram of Garson, Energy & Carbon Audits). Johannes is preparing Sankey Diagrams of energy and carbon consumption for Deep Nickel Mines.

B.Eng. Civil Engineering, Ana Leite (Energy efficiency of generic comminution processes). Investigated the multiplicity of ways in which large rocks are reduced to small rocks (blasting, hydrofracture, grinding, etc.) and considered these processes from the point of view of both energy intensity, and power intensity. A surprise finding was that the power produced in a typical development blast was of the order of one GW. When power and energy intensity were normalized by the surface area introduced in the size reduction process, blasting was identified as one of the most energy and power efficient process (Figure 110), which begged the question, why do we not see explosive based crushing, grinding and drilling machines?

Mr Lewis Oatway, Vale. Energy Superintendent. Supplied detailed data on all fuels consumption for assets within the Sudbury Basin as well as 15 second time series data on total electricity consumption for Garson Mine.

Selected References

Levesque, M. & Millar, D.L., 2014. The link between operational practices and specific energy consumption in metal ore milling plants – Ontario experiences. Minerals Engineering. Vol. 71, p.146-158. Nominated for the CEEC Medal.

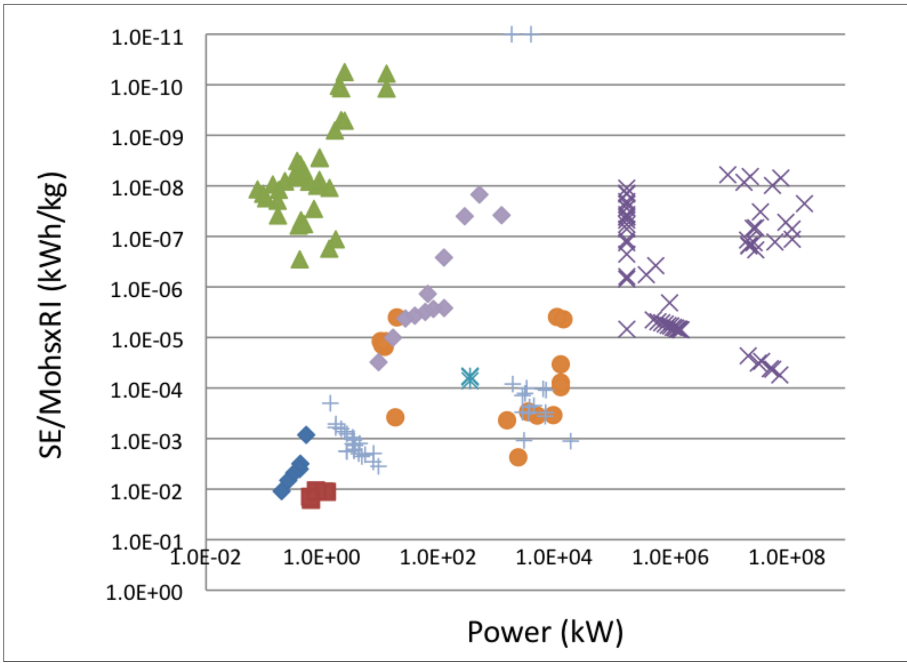


Figure 110: Energy consumed in rock comminution plotted against the power of comminution for varying methods of comminution

6.4 Auxiliary Ventilation Systems

The SUMIT 6 program was designed specifically to follow on from Ventilation-On-Demand work that was coordinated by CEMI before the start of the SUMIT Program. Data sets investigated by SUMIT 6 researchers suffered from incompleteness which made analysis to establish definitive answers very challenging. However, during the course of these investigations, the characteristics of ducts to which fans, whether they be variable speed or not, were connected was found to be equally important in the energy saving equation.

Methodology

Practical field investigations were conducted which aimed to assess the roughness of varying duct types (Figure 111).

Figure 111: The yellow sections of the auxiliary ventilation system pictured above are novel low friction plastic ducting segments that are supplied by Gplus Industrial Plastics Inc..
Source: www.plastiquesgplus.com/en/ventilation

Summary of Findings

It became apparent that leakage of auxiliary ventilation duct systems was equally, if not more important, than roughness, and a leaky duct system could not only defeat the energy savings intentions of a VOD strategy, but could lead to more energy being consumed not less, if a minimum amount of air flow had to be delivered to a working face to support a specific level of production (Figure 112).

Simulation analyses were extended to establish how the electrical energy consumption, and the air flow delivered to the working face, vary as the auxiliary ventilation duct lengthens (Figure 113). Application of smart technologies that discriminate between two distinct types of the resulting power curve may permit classification of whether a duct may be deemed to be leaky or not.

Varying the speed of a fan driving an auxiliary ventilation system (as in VOD) is not the only way in which such ventilation systems can be controlled with an objective of minimising ventilation cost. The resistance to which the ventilation fan is connected can be adjusted too. In the case of passive regulators, the energy in the air flow that becomes surplus to requirements is simply dissipated in frictional and shock resistance through the regulator.

Duct Roughness		Leakage		
		Very low R_j = 10,000,000 Ns^{2}m^{-8}	Medium R_j = 200,000 Ns^{2}m^{-8}	High R_j = 10,000 Ns^{2}m^{-8}
Smooth k = 0.0018 (kg/m^3)	N fan	1780 rpm	1780 rpm	1780 rpm
	Q fan	39.470 m^3/s	40.728 m^3/s	43.589 m^3/s
	Q face	38.580 m^3/s	34.914 m^3/s	24.040 m^3/s
	Fan kW	114.6 kW	113.26 kW	106.7 kW
	COV	3.0 kW/m^3/s	3.2 kW/m^3/s	4.4 kW/m^3/s
	L%/100m	0.9 %	5.9 %	18.6 %
	k US	-2.4 %	-14.54 %	-41.9 %
	k DS	2.0 %	14.6 %	80.2 %
	k AVE	-0.2 %	-1.5 %	-5.4 %
Medium k = 0.0021 (kg/m^3)	N fan	1780 rpm	1780 rpm	1780 rpm
	Q fan	38.551 m^3/s	39.948 m^3/s	43.102 m^3/s
	Q face	37.644 m^3/s	34.019 m^3/s	23.119 m^3/s
	Fan kW	115.4 kW	114.1 kW	108.3 kW
	COV	3.1 kW/m^3/s	3.4 kW/m^3/s	4.7 kW/m^3/s
	L%/100m	1.0 %	6.2 %	19.2 %
	k US	-2.5 %	-15.1 %	-43.2 %
	k DS	2.0 %	15.2 %	84.5 %
	k AVE	-0.3 %	-1.7 %	-6.0 %
Rough k = 0.0037 (kg/m^3)	N fan	1780 rpm	1780 rpm	1780 rpm
	Q fan	34.135 m^3/s	36.240 m^3/s	40.836 m^3/s
	Q face	33.175 m^3/s	29.925 m^3/s	19.684 m^3/s
	Fan kW	118.6 kW	117.0 kW	113.0 kW
	COV	3.6 kW/m^3/s	3.9 kW/m^3/s	5.7 kW/m^3/s
	L%/100m	1.2 %	7.2 %	21.5 %
	k US	-3.0 %	-18.0 %	-48.6 %
	k DS	2.4 %	18.1 %	106.2 %
	k AVE	-0.4 %	-2.4 %	-8.6 %

Figure 112 Tabulated results of simulations of auxiliary ventilation duct comprising a fixed number of equal length segments, with varying levels of leakage and duct roughness for a constant speed fan N fan = 1780rpm. Low leakage (well coupled together), smooth ducts are best, having the lowest Cost Of Ventilation (COV), and the lowest loss per 100 m of duct length (L%/100m). When determining duct friction factor, the best agreement between known and measured roughness coefficients occurs when two flow rate observations are taken: one upstream of the section under test (US) and one downstream (DS), and the flow rates are then averaged

As an alternative to this, the surplus energy in the air flow can be harnessed and used to provide ventilation network resistance through the recovery of electricity. A parallel aerofoil type active air flow regulator was designed and tested as part of the SUMIT 6 program. As well as providing a further energy saving option for control of mine ventilation networks, this device could be operated as a stand-alone wind turbine.

Team Contributions

PhD. Natural Resources Engineering, Michelle Levesque undertook field observations and techno-economic simulations of auxiliary ventilation systems. Dean prepared a simulation tool that specifically accounted for leakage in these systems. Michelle and Dean found that leakage, not duct roughness, nor ventilation-on-demand, was the dominant factor governing the performance of auxiliary ventilation systems in mines.

PhD. Natural Resources Engineering, Harvard Farrant (expected submission Dec 2016, Aerodynamics of a Novel MBWT Rotor). Harvard has made valuable research presentations on both the MESCO (mine energy services companies) concept and his PhD research, but has not published much on his PhD topic due to its potentially proprietary, commercializable nature. During Quarter 2, 2015, supported by SUMIT, Harvard submitted a Canadian Patent application document to obtain a priority date for the intellectual property in the design of his wind turbine / dynamic air flow regulator rotor (Figure 114).

B.Eng. Mining Engineering, Felipe Gabriel (Dynamic ventilation network modelling). Felipe examined the dynamics of auxiliary ventilation systems, with a particular view to providing enhanced air flow for contamination dilution and clearance.

Selected References

Millar, D.L., Levesque, M., and Hardcastle, S., 2016. Leakage and air flow resistance in mine auxiliary ventilation ducts: effects on system performance and cost. Transactions of the Institute of Mining and Metallurgy, Part A: Mining Technology. Mining Technology, DOI: 10.1080/14749009.2016.1199182.

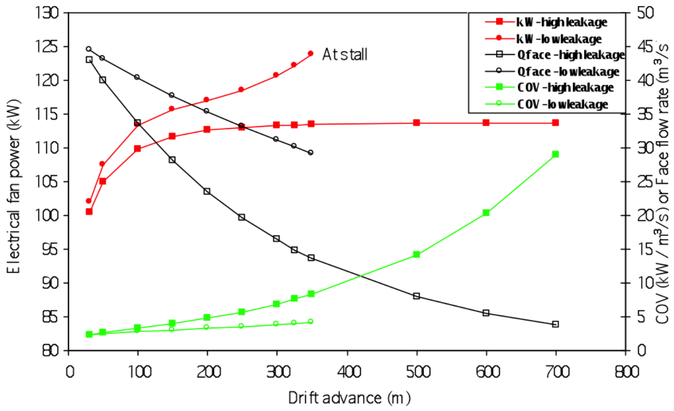


Figure 113: Power consumed by fan, air flow delivered to a working face and cost of ventilation (COV) for duct cases of equal roughness but either leaky or non-leaky couplings

Software Development Team Contributions

M.Sc. Computer Science, Adam Turcotte (Dec 2012, Anti-aliasing image re-samplers). Adam joined the SUMIT 6 team to provide dedicated computer programming support to the team. At the same time as completing his masters, he led on the development of the 4D Sankey Trees software product which is available for download via the MIRARCO website.

PhD. Computer Science, Tim Doan (ongoing, Algorithms to enhance virtual reality environments). Tim is a graduate student at Laurentian University reading Computer Science. His work has been supported by additional funds from the SUMIT Program.

This regulator dissipates air power

This regulator extracts power from the mine air flow and turns it into electricity

Figure 114: An air flow regulator that provides resistance by recovering air flow energy by using moving blading to drive an electrical generator

6.5 Integrated Technologies

The SUMIT 6 award to Dr. Millar was pivotal in the establishment of a new research group within MIRARCO, the Energy Renewables and Carbon Management group.

Methodology

Leveraged human capacity permitted investigation of allied energy technologies that could be valuable to the Canadian and Ontario mining industry (Figure 115). Prominent among these were technologies that could lead to novel concepts for Deep Mine Cooling, and within this group of technologies, the most important of these was the investigation of hydraulic air compressors undertaken by Dr. Millar.

Summary of Findings

The research investigations by Dr. Millar led to the submission of patent applications, the establishment of a start-up technology company, Electrale Innovation Ltd, and a follow-on award from the Ultra Deep Mine Network, a Network Centre of Excellence, to establish a hydraulic air compressor demonstrator facility.

Team Contributions

M.Sc. Nuclear and Renewable Energy Engineering: Valeria Pavese (May 2015, Mechanical Efficiency of Hydraulic Air Compressors). Valeria worked on checking solubility and hydrodynamic calculations on Hydraulic Air Compressors.

Other Contributions

B.Eng. Chemical Engineering, Julia Andrade (Gas solubility calculations for hydraulic air compressors). Julia's work lead to a realisation that Hydraulic Air Compressors could be a energy efficient technology for carbon capture.

B.Eng. Chemical Engineering, Eden Laurindo (Modeling and manufacture of a 'snail shell' hydrodynamic compressor). Eden designed, CFD modeled and fabricated a compact, extra long diffuser venturi eductor that took the form of a snail shell.

B.Eng. Control and Instrumentation Engineering, Diogo Sanchez de Oliveira (Instrumentation and control design for Baby HAC). Diogo specified the control and instrumentation scheme for the 'Baby HAC' pilot scale hydraulic air compressor.

B.Eng. Mining Engineering. Flavia Vespucio and B.Eng. Mining Engineering. Angelina Rocha. They worked together on the challenging job of undertaking CAD and CFD modelling of the Taylor head of the Hydraulic Air Compressor at Peterborough Lift Lock.

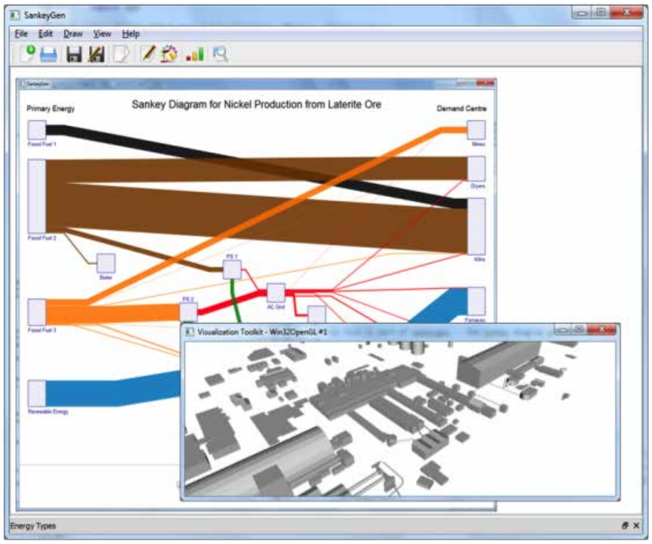


Figure 115: Integrated assessment and visualization tools

B.Eng. Mining Engineering. Research Project Students (2013)

- – Justin Bontin (Modeling and experimental design of an air water separator)
- – Jordan Gladu (Modeling and experimental design of an air water separator)
- – Alex Hutchison (Modeling and experimental design of an air water separator)
- – Gabriel Janakavaj (Modeling and experimental design of an air water separator)

Selected References

Pavese, V., Millar, D., Verda, V., 2016. Mechanical efficiency of hydraulic air compressors. ASME Journal of Energy Resources Technology. doi:10.1115/1.4033623.

Millar, D.L., 2014. A review of the case for modern-day adoption of hydraulic air compressors. Applied Thermal Engineering. Vol. 69, No 1-2, p.55-77.

NOTE: for a complete listing of the principal author's publications please go to: http://www.mirarco.org/ercm/

SUMIT PROGRAM OVERALL CONTRIBUTIONS

General

The SUMIT Program has demonstrated the value of effective coordination of a collaborative multi-institutional mining-related R&D program. It can be a model for future potential multi-stakeholder programs of this kind.

One third financial and in kind contributions by all stakeholders, industry, the provincial government, and institutions, has lead to the attainment of virtually all goals set for the program – whether HQP-training, technical or other measures of success.

HQP Development

Figure 116 is an overall summary of the metrics related to personnel training, publications produced and citations received[7], presentations made, hours of fieldwork conducted and post-program jobs creation for students.

Further, it captures the very important measure of safe performance of multiple underground field programs. This is a testament to the careful supervision and effective training provided by designated representatives of each of the company test sites used.

Technical

The SUMIT Program concept grew out of an initiative created by CEMI to address the problem of fault-slip-related rockbursts in underground mines but was later altered somewhat to match available provincial government funding options, namely to include ICT and energy optimization-related work. It nevertheless, at its core, facilitated important R&D on rock mass characterization, change detection, tools and other software development which serve to shed considerable light on the fault-slip problem.

As a bonus, considerable advances were also made in developing novel approaches to large dataset sharing and management as well as to finding optimal approaches for energy management in mines.

The principle technical solutions of the program have already been summarized in this document (pp v-vi) and won't be further elaborated upon here.

SUMIT ACHIEVEMENTS TO DATE

98 participants

83 conference presentations

citations in scientific and engineering literature **374**

206 publications

student hours of safe in-mine work **959**

0 incidents or accidents

new positions for SUMIT student graduates **19**

SUMIT HQP DEVELOPMENT

75	students
7	post doctoral fellows
19	doctoral students
30	master's students
19	undergraduate students
17	researchers
1	research engineer

Figure 116: Major metrics related to the outcomes of the SUMIT Program

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[7] *At time of writing (Q1, 2017)*

Appendix

APPENDIX 1

LINKS TO SUMIT COMMERCIALIZED PRODUCTS

1. Geoscience Integrator
a product of Mira Geoscience
www.mirageoscience.com/our-products/software-product/geoscience-integrator

2. BurstSupport Tool
a product of MIRARCO Mining Innovation
www.mirarco.org/burstsupport/

APPENDIX 2

A GROWTH STRATEGY FOR THE MODCC

MINING OBSERVATORY DATA CONTROL CENTRE

GOAL

Design a roadmap for the future which clearly identifies the directions in which MODCC needs to go to provide maximum benefit to the Canadian mining industry.

Resolution mining office at MODCC. Photo credit: Damien Duff

CONTEXT?

"Mining Big data" has become synonymous with the need for in-dustry and academia to make better use of the myriad datasets it gathers. Some 67% of mining company executives polled have suggested that they see deriving value from their datasets as being key to their company's survival and prosperity.

MODCC was established to ensure that CEMI was at the forefront of new developments in this area and, through what has already been accomplished as an incubator space, CEMI has managed to generate significant interest. Our involvement with Mira Geoscience, seen as a key enabler of exploration and mining data integration for new ore deposit discovery and geotechnical hazard assessment, has meant that we already are teamed with a reputable and motivated collaborator.

CURRENT STATUS OF MODCC

Three incubates are resident at MODCC; Mira Geoscience, Rev-olution Mining, and Tunik Inc. The application material from a fourth – SCICOM – is under review presently.

MODCC is also home to prepared datasets from SUMIT and the CMIC Footprints projects. These datasets have been formatted and inputted into Mira Geoscience's INTEGRATOR platform. They represent an excellent opportunity for further research using real mine data for academic and private sector researchers.

SUGGESTED PATH GOING FORWARD

A two-pronged approach is advocated (Figure 117). It will allow for growth and the firm establishment of MODCC as the "goto" place for exploration/mining data analytics capability in this part of the world.

a) Support Academia

For academic research projects, MODCC represents an optimal place to centralize cleaned datasets. The venue, plus Mira Geoscience INTEGRATOR software access, allows other researchers to parse and reparse information and rapidly accelerate progress of new discoveries.

b) Support Mining Companies

CEMI has begun engaging with mining companies for digital demonstration projects. Leveraging innovative Canadian Smallto-Medium Enterprises (SMEs) CEMI is building consortiums that solve important challenges related to exploration, mining productivity, geotechnical risk, energy optimization, and environmental issues with digital technologies. The by-product of these solutions are clean datasets that can attract additional innovation.

c) Supporting Canadian SMEs)

Through the MODCC incubation space, demonstration projects, access to data, funding and commercialization support, CEMI seeks to accelerate and enhance the ability for innovation SMEs to bring technology to market. CEMI will also continue to support SME prosperity in Canada by connecting mining industry SMEs with other Canadian digital ecosystem partners and SMEs who can become partners in technology development.[8]

For more info contact: Marcus Thomson at mthomson@cemi.ca
www.cemi.ca

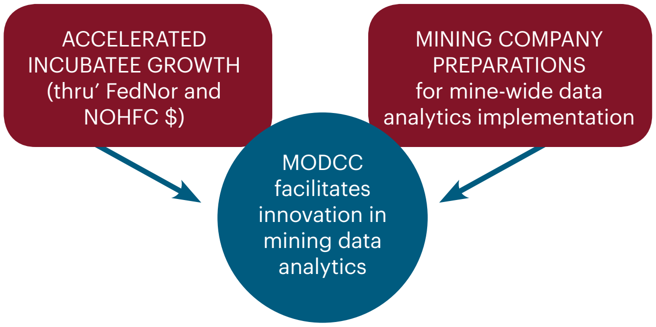

Figure 117: Facilitating innovation in Mining Data Analytics through a two-pronged approach

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[8] *This will likely include working with companies to make datasets available to researchers through MODCC.*

APPENDIX 3

Vol. 4: November, 2014

SUMIT - Smart Underground Monitoring and Integrated Technologies for deep mines

FURTHER NEWS FROM "THE SUMIT"

- **Program has now entered its 4th of 5 years**
- **2 Sub-projects have been completed and a further 3 sub-projects are scheduled to be completed in 2014.**
- **$5.28M of $6.70M has been expended on research activities**

SUMMARY

The SUMIT project continues to progress on schedule and to meet its predetermined milestones and deliverables. Most research projects will be ending in 2015, however some will extend into 2016. All research must be completed by June 30th, 2016 and the program wrapped up by Mar. 31st, 2017.

The number of involved researchers remains at 15 whereas the total student involvement has decreased from 41 to 40 (1 PDF; 15 PhD; 16 MSc and 8 undergraduate). In addition, 4 other individuals are involved as researcher engineers etc). One PhD, 6 MSc. and 4 SUMIT-involved/supported students have been trained and have graduated to date.

Field work involving UofT and Queen's teams is proceeding at Creighton and Nickel Rim South mines in Sudbury.

At time of writing, records show that some 17 technical articles have been published in peer-reviewed journals with a further 3 accepted for publication and 16 submitted. Eighteen refereed conference papers have been produced and some 37 conference presentations/posters made.

ONGOING COLLABORATIONS

Design of the Mining Observatory Data Control Centre (MODCC), located at the surface facility of SNOLab in Sudbury, has been completed. Tenders for construction are now being requested. Completion of the facility is scheduled during late Q1, 2014.

Discussions have been initiated with a like-minded data management group out of the University of Arizona (MINA- Mining Intelligence Network of the Americas) and possible synergistic approaches are being investigated.

Experimental work related to SUMIT project 4B-(i) *"Preconditioning naturally fractured hard rocks using hydraulic fracturing"* at an underground mine owned by Newcrest Mining Limited in NSW Australia has received a set-back and preliminary discussions are now underway with a research and alternative test-site ownership group in Switzerland to complete the work.

HEALTH & SAFETY

We continue to remain very pleased to report that no serious injuries or accidents have been recorded to date.

- 6 sub-projects in 2 mines
- 753.5 hours
- 16 researchers from 5 universities

CEMI 935 Ramsey Lake Road, Willet Green Miller Centre, Sudbury, Ontario P3E 2C6
www.miningexcellence.ca or email us at info@miningexcellence.ca

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DeepMine

COMMUNICATIONS AND OUTREACH

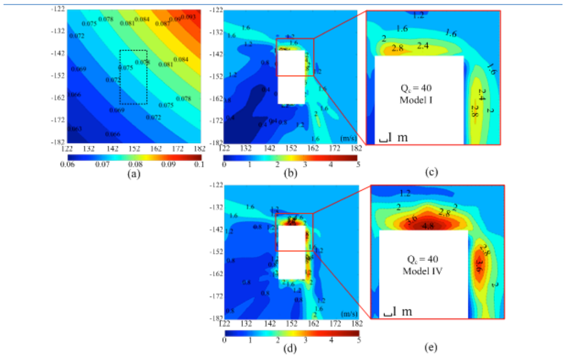
CEMI is currently developing a SUMIT video series to engage youth and showcase the goals and milestones of the SUMIT project. The video project consists of a series of lecture videos featuring students from the SUMIT project. The videos are being shot throughout the fall of 2014 and into the spring of 2015.

BUDGET, FUNDING AND LEVERAGING

Total project expenses to date are $5.28M.
Some interest has been expressed by SUMIT researchers in leveraging their sub-project funds through the Ultra Deep Mining Network (UDMN) at CEMI and LOI's have been submitted for review.

DELIVERABLES DURING THE REPORTING PERIOD (Q2'14 – Q3'14)

1. Development of a non-uniform velocity model which considers confinement and rockmass failure to assist with simulation of wave field propagation around excavation openings and thereby potentially identify zones at a higher risk for rockbursts.



Site amplification of the stope. (a) PPV contour without excavation, which is based on the design scaling law with a 90-95% confidence level; (b) and (d) amplification factor contours around the stope and zoom-in plots (c) and (e) showing the detailed site amplification in the roof and on the right wall for uniform and non-uniform velocity models, respectively.

2. Commencement of work to investigate how the use of LiDAR measurements of drift deformation can be used to infer information about the stress field at the drift scale.
3. Continued development of stress monitoring technology through the integration of petrophysical, borehole geophysical and geotechnical data.

NEXT STEPS

1. Scheduling of Third Annual SUMIT Stakeholder Update Meeting— likely in April, 2015 at the University of Waterloo.
2. Continuation of extensive field work activities Q4, 2014.

FOR FURTHER INFORMATION

Please contact Damien Duff at:
dduff@miningexcellence.ca

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in Mining Innovation

935 Ramsey Lake Road, Willet Green Miller Centre
Sudbury, Ontario, Canada P3E 2C6

T: 705.673.6568 | F: 705.671.3878
info@cemi.ca | www.cemi.ca